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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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CUT-AND-FILL STOPING

BY

C. H. JOHNSON AND E. D. GARDNER

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By C. H. Johnson[2] and E. D. Gardner[3]

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6688."

2 - Assistant mining engineer, U. S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

3 - Supervising engineer, U. S. Bureau of Mines, Southwest Experiment Station, Tucson, Ariz.

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INTRODUCTION AND ACKNOWLEDGMENTS

This paper on the cut-and-fill method of mining is one of a series on mining methods published by the United States Bureau of Mines. It has been compiled chiefly from recent publications of the bureau describing in detail mining methods and costs at various mines. Other sources of information were the technical press, the Transactions of the American Institute of Mining and Metallurgical Engineers, the Mines Handbook, and company reports. References other than those mentioned in Table 1, which lists the mines entering into this discussion and gives general data regarding them, are given in the body of the paper. The authors wish to acknowledge the kind assistance of many operating officials, either through correspondence or in the course of numerous visits to their mines.

DEFINITION OF CUT-AND-FILL STOPING

The cut-and-fill method of stoping is that method in which relatively small sections or slices are taken successively from the lower surface of a block of ground by open-stoping methods, the broken ore being removed and each section filled with waste material before the next is mined. This definition excludes shrinkage stoping, even with subsequent filling; it excludes square-set and fill methods; it excludes the Mitchell slice method, which may be classed as underhand square-setting; it excludes the systematic use of stull sets and filling, which is a variation of square-setting; it excludes any method involving the caving of the ore, walls, or overburden during the process of extraction, although caving may be allowed to take place when the waste fill is withdrawn for use in other stopes; and it excludes any open-stope method without filling. Through variations, which will be discussed later, the cut-and-fill method merges into shrinkage stoping, open stoping, or square-setting. The use of casual timber support, such as props, stulls, or timber cribs for holding up weak portions of the back does not constitute a change from cut-and-fill to square-set mining.

Cut-and-fill stoping is divided into two submethods: The horizontal cut-and-fill, or flat-back method, and the inclined cut-and-fill, or filled-rill method. In the former the back-and-fill surfaces are kept practically horizontal. In the latter the back and fill are maintained parallel to each other, but at a considerable angle with the horizontal, usually such that the most of the broken ore will run by gravity down the slope to the extraction chute and the fill material will run into the stope and lie at its angle of repose.

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Table 1.- Location, production, and general data on mines using cut-and-fill methods, with partial bibliography

Mine	Company	Location	Chief metals	Approximate annual tonnage	Mining methods
1	2	3	4	5	6
[1]	Arizona Copper Co.	Morenci, Ariz.	Copper	1,200,000	Many, including cut-and-fill.
[2]	Calumet & Arizona Mining Co.	Warren, Ariz.	Copper	500,000	Rill; semishrinkage; square-set; Mitchell slice.
[3]	Iron Cap Copper Co.	Globe, Ariz.	Copper	90,000	Rill.
[4]	Magma Copper Co.	Superior, Ariz.	Copper	250,000	(Rill, 35 per cent (est.) (Mitchell slice, 40 do. do. (Square-set, 25 do. do.
[5]	Phelps Dodge Corporation	Bisbee, Ariz.	Copper	[6]400,000	(Square-set, 55 do. (Top-slice, 40 do. (Cut-and-fill, 4 do.
[7]	United Eastern Mining Co.	Oatman, Ariz.	Gold	100,000	Horizontal cut-and-fill.
[8]	United Verde Copper Co.	Jerome, Ariz.	Copper	1,000,000	(Cut-and-fill, 61 per cent (Rill, 3 do. (Square-set, 27 do. (Top-slice, 8 do. (Shrinkage, 1 do.
[9]	Liberty Bell Gold Mining Co.	Telluride, Colo.	Gold-silver	130,000	(Open stulled stopes. (Formerly cut-and-fill.
——[10]	——	Telluride district, Colo.	Gold-silver	——	Various, including cut-and-fill.
[11]	Bunker Hill & Sullivan Mining & Concentrating Co.	Kellogg, Idaho	Lead-silver	450,000	(Square-set. (Some cut-and-fill.

[1] Scotland, P. B., Mining Methods of the Arizona Copper Co.: Trans. Am. Inst. Min. Met. Eng., vol. 51, 1915, pp. 267-280.

[2] Lavender, H. M., Mining Methods of the Campbell Mine of the Calumet & Arizona Mining Co., Warren, Ariz.: Inf. Circ. 6289, Bureau of Mines, 1930, 18 pp.

[3] Lees, C. E., Mining Methods and Costs at the Iron Cap Copper Co., Copper Hill, Ariz.: Trans. Am. Inst. Min. Met. Eng., vol. 72, 1925, pp. 371-380.

[4] Snow, F. W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Inf. Circ. 6168, Bureau of Mines, 1929, 32 pp.

[5] Oct. 27, 1928, issue of Eng. and Min. Jour., featuring the operations of the Phelps Dodge Corporation.
Fullen, J. B., Resume of History and Operations of the Copper Queen Branch of the Phelps Dodge Corporation: Min. Congress Jour., Oct., 1930, pp. 764-772.

[6] From company reports, excludes porphyry ore production.

[7] Moore, R. W., Mining Methods and Records at the United Eastern Mine: Tech. Publication 83, Am. Inst. Min. Met. Eng., March, 1928, 35 pp.

[8] Quayle, T. W., Mining Methods and Practices at the United Verde Copper Mine, Jerome, Ariz.: Inf Circ. 6440, Bureau of Mines, 1931, 31 pp.
April, 1930, issue of the Min. Cong. Jour., featuring the operations of the United Verde Copper Co.

[9] Chase, C. A., Notes on the Liberty Bell Mine: Trans. Am. Inst. Min. Met. Eng., vol. 42, 1912, pp. 694-741.

[10] Bell, C. N., Mining Methods of the Telluride District: Trans. Am. Inst. Min. Met. Eng., vol. 72, 1925, pp. 550-565.

[11] Brown, U. E., Mining Methods of the Bunker Hill and Sullivan Mining and Concentrating Co., Kellogg, Idaho: Inf. Circ. 6407, Bureau of Mines, 1931, 9 pp.

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Table 1. Location, production, and general data on mines using cut-and-fill methods, with partial bibliography-Conti

Mine	Company	Location	Chief metals	Approximate annual tonnage	Mining methods
1	2	3	4	5	6
Various[12]	Copper Range Co.	Michigan	Copper	500,000	Horizontal cut and fill.
Various[13]	Anaconda Copper Mining Co.	Butte, Mont.	Copper	3,000,000	Square set, 80 per c
Pecos[14]	American Metal Co. of New Mexico	Tererro, N. Mex.	Zinc lead copper	200,000	(Hor.cut and fill, 40 do (Square set, 50 d
Ground Hog[15]	Asarco Mining Co. (A. S. & R.)	Vanadium, N. Mex.	Lead zinc copper	40,000	Square set; rill; some hori cut and fill.
Eighty-Five[16]	Calumet & Arizona Mining Co.	Valedon, N. Mex.	Copper	90,000	(Shrinkage, 85 per cent (es (Rill, 15 do. d
Questa[17]	Molybdenum Corporation of America	Questa, N. Mex.	Molybdenum	10,000	(Hor.cut and fill, 90 per c (Open stope, 10 do
Park Utah[18]	Park Utah Consolidated Mines Co.	Park City, Utah	Silver lead zinc	180,000	Square set; cut and fill; stulled stopes.
----[19]	-----	Broken Hill, N.S. W., Australia	Lead-silver-zinc	----	Various, including horizont cut and fill.
Britannia[20]	Britannia Mining & Smelting Co. (Howe Sound Co.)	British Columbia, Canada	Copper	1,000,000	Shrinkage; square set; rill stoping.
Frood[21]	International Nickel Co. Co.	Sudbury, Ontario, Canada.	Copper nickel	1,200,000	Horizontal cut and fill; so square set.
Matahambre[22]	American Metal Co.,(Ltd.)	Matahambre, Pinar del Rio, Cuba	Copper	350,000	(Hor.cut and fill, 70 per (Square set, 30 do

12 – Crane, W. R., Mining Methods and Practice in the Michigan Copper Mines: Bull. 306, Bureau of Mines, 1929, 192 pp. Schacht, W. H., Mining Methods of the Copper Range Co.: Trans. Am. Inst. Min. Met. Eng., vol. 72, 1925, p 370. Mendelsohn, Albert, Mining Methods and Costs at the Cnampion Copper Mine, Painesdale, Mich.: Inf. Circ. 6515, of Mines, 1931, 16 pp.

13 – Daly, W. B., and others, Mining Methods in the Butte District: Trans. Am. Inst. Min. Met. Eng., vol. 72, 1925, pp. 23 Dunshee, B. H., Timbering in the Butte Mines: Trans. Am. Inst. Min. Met. Eng., vol. 46, 1914, p. 147. Rahilly, H. J. Fires and Hydraulic Filling: Trans. Am. Inst. Min. Met. Eng., vol. 68, 1922, pp. 61–72. Bicknell, H. L., Rill Stop the Butte District: Compressed Air Mag., vol. 26, July, 1921, pp. 10153–10058.

14 – Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ Bureau of Mines, 1930, 21 pp.

15 – Richard, F. W., Mining Methods and Costs at the Ground Hog Unit, Asarco Mining Co., Vanadium, New Mexico: Inf. Cir Bureau of Mines, 1930, 13 pp.

16 – Youtz, R. B., Mining Methods at the Eighty Five Mine, Calumet and Arizona Mining Co., Valedon, N. Mex.: Inf. Cir Bureau of Mines, 1931, 26 pp.

17 – Carman, J. B., Mining Methods of the Molybdenum Corporation of America at Questa, N. Mex.: Inf. Circ. 6514, Bureau o 1931, 15 pp.

18 – Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Inf. Circ. 6290, Bureau of Mines 18 pp.

19 – Horwood, E. J., Broken Hill Underground Mining Methods: Trans. Am. Inst. Min. Met. Eng., vol. 54, 1917, pp. 54–73.

20 – Moore, J. I., Operations at the Britannia Mines: Eng. Min. Jour., vol. 122, Dec. 11, 1926, pp. 924–930.

21 – Mutz, H. J., Mining the Frood Orebody at Depth: Eng. Min. Jour., vol. 130, Nov. 10, 1930, pp. 445–452.

22 – Richert, G. L., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Inf. Circ. 6145, Bureau of Mines, 1929 Richert, G. L., Filling Stopes with Mill Tailing at Minas de Matahambre, in Pilar del Rio, Cuba: Eng. Min. Jour. 127, March 2, 1929, pp. 348–352.

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Table 1.- Location, production, and general data on mines using cut-and-fill methods, with partial bibliography-Concluded

Mine	Company	Location	Chief metals	Approximate annual tonnage	Mining methods
1	2	3	4	5	6
aruma[23]	South American Development Co.	Zaruma, Ecuador	Gold	90,000	Hor. cut-and-fill; rill.
ro General[24]	Cia. Minera Asarco (A. S. & R.)	Charcas, San Luis Potosi, Mexico	Zinc-lead	(?)	Underhand cut-and-fill.
olorada[25]	Cananea Consolidated Copper Co.	Cananea, Sonora, Mexico	Copper	500,000	Rill, 40 per cent Hor. cut-and-fill, 14 do. Square-set, 12 do. Shrinkage, 14 do. Drifting, 20 do.
ilares[26]	Moctezuma Copper Co. (Phelps Dodge Corporation)	Pilares de Nacozari, Sonora, Mexico	Copper	800,000	Hor. cut-and-fill, 60 per cent Rill, 18 do. Square-set, 18 do. Top-slice, 3 do. Shrinkage, 1 do.
anta Gertrudis[27]	Cia. de Santa Gertrudis	Pachuca, Hildago, Mexico	Silver-gold	(?)300,000	Horizontal cut-and-fill; square-set; shrinkage.
cky Tiger[28]	Tigre Mining Co.	El Tigre, Sonora, Mexico	Silver-gold	70,000	Hor. and rill cut-and-fill. Some shrinkage and open stoping.
——[29]	Syndicate Mining Co.	Rio. Guinobatan, P. I.	Gold	(?) 40,000	Cut-and-fill.
——[30]	———	Krivoy Rog District, Russia	Iron	6,500,000 (1913)	Open stopes; cut-and-fill; shrinkage.

3 - Ezzel, Rudolph, Mining Methods in Zaruma District, Ecuador: Trans. Am. Inst. Min. Met. Eng., vol. 72, 1925, pp. 447-467.

4 - Willey, Howard, Top Slicing with Filling of Slices, as Used at the Charcas Unit of the Cia. Minera Asarco, S. A.: Tech. Publication 364, Am. Inst. Min. Met. Eng., Sept., 1930, 17 pp.

5 - Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Inf. Circ. 6247, Bureau of Mines, 1930, 41 pp.

Elsing, M. J., Mining Methods Employed at Cananea, Mexico: Eng. Min. Jour., vol. 90, Nov. 5, 1910, p. 914, and vol. 90, Nov. 12, 1910, p. 963.

6 - Leland, Everard, Mining Methods and Costs at the Pilares Mine, Pilares de Nacozari, Sonora, Mexico: Inf. Circ. 6307, Bureau of Mines, 1930, 34 pp.

Wade, W. R., and Wandke, Alfred, Geology and Mining Methods at Pilares Mine: Trans. Am. Inst. Min. Met. Eng., vol. 63, 1920, pp. 382-407.

7 - Rose, Hugh, Mining and Milling Practice at Santa Gertrudis: Trans. Am. Inst. Min. Met. Eng., vol. 55, 1917, pp. 397-435.

8 - Mishler, R. T., and Budrow, L. R., Methods of Mining and Ore Estimation at the Lucky Tiger Mine: Trans. Am. Inst. Min. Met. Eng., vol. 72, 1925, p. 468.

9 - Kurie, F. M., Operating a Gold Property in the Philippine Islands, Eng. Min. Jour., vol. 128, Dec. 28, 1929, pp. 1004-1005.

0 - Bouldovsky, A. K. (translated from the original Russian by W. Ayvazoglou), The Methods of Underground Mining of Iron Ore in the District of Krivoy Rog: Inf. Circ. 6254, Bureau of Mines, 1930, 48 pp.

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HISTORY OF CUT-AND-FILL MINING

The cut-and-fill method of mining was an old and well-established one in the mines of Europe at least previous to the publication of Callon's lectures on mining, which were translated into English about 1876. Probably the first extensive use of the method in the United States was in 1884, in the Chapin iron mine at Iron Mountain, Mich.[4] It was introduced there to supplant a stope-and-pillar method. The stopes were supported by very heavy timbering as well as by the pillars of ore, but as the mine became larger and deeper, the pillars began to fail, although they had been increased in size until they constituted 50 per cent of the ore. By the new method, ore which was moderately firm and not over 20 feet wide was mined by taking overhand cuts along the strike of the vein, followed by filling with waste from the surface. In wider ore-bodies cuts 8 feet high and 9 feet wide were taken from foot to hanging wall, followed by waste filling, and subsequent cuts alongside, until a complete floor was extracted, whereupon the process was repeated on the next floor above. Some of the old crushed pillars were extracted also in this manner.

In a few years the cut-and-fill method was adopted in practically all of the mines on the Menominee range.[5] In many of these mines it was supplanted in turn by caving methods. Apparently the new method did not come into use in western mines until between 1905 and 1910. Since then the method has been used in many of the mining districts of the United States and other countries in many forms and under varying conditions.

EXPLORATION

Maximum production and efficiency in the mining of any orebody depend partly on having complete knowledge of its size, shape, and character before mining begins. While this is true regardless of the mining method used, the importance of this factor varies. In square-set stoping, for instance, prospecting of the walls can be carried on, extensions of the ore into the walls can be mined, waste inclusions can be avoided or mined separately, and ore and waste can be handled with relatively little regard for the location of chutes and haulage drifts and without serious effects on production or future mining. In shrinkage stoping this is not the case, as satisfactory results are obtained only by systematic breaking and drawing. If drawing must be stopped in one part of a shrinkage stope in order to extract an ore extension into the wall, the ore may be drawn too fast from other parts of the stope, or the concentration of mining operations in another part of the stope necessary to maintain production may cause the outline of the back to become inconveniently irregular. The presence of an unsuspected body of waste will greatly hamper operations in a shrinkage stope, unless it is of such nature that it can be left in place as a pillar. Shrinkage stoping can not readily be changed to other methods; it must therefore be considered as a relatively inflexible method which demands complete knowledge of the physical character of the orebody previous to mining.

Cut-and-fill mining lies between square-setting and shrinkage stoping in respect to flexibility, and correspondingly with respect to the amount of exploration required. The highest efficiency can be obtained only with proper location of chutes, manways, haulage ways, and waste raises, but the results of improper location are not as serious as in shrinkage stoping in which it often happens that parts of the orebody may be completely out of reach of the original layout of drifts and chutes. Low-grade ore left behind in a cut-and-fill stope is less readily mined at a later time than if square-setting had been used, but

4 Larsen, Per, The Chapin Iron Mine, Lake Superior: Trans. Am. Inst. Min. Eng., vol. 16, 1887, p. 119.

5 Fulton, John, Methods of Mining in the Menominee Range, Michigan: Trans. Am. Inst. Min. Eng., vol. 16, 1887, p. 891.

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if proper precautions have been taken in regard to gob fencing, such ore can later be more safely extracted than from the walls of an old shrinkage stope. As in square-setting, prospecting of the walls of a horizontal cut-and-fill stope can be done without much interference with stoping. Ore extensions or waste inclusions present no serious obstacles. If the character of the ground makes it desirable, the method is changed readily to shrinkage or square-set stoping.

Drifting in the ore is the method generally used to explore an orebody previous to cut-and-fill stoping. If the orebody is of the vein type, is regular, and distinctly outlined, and particularly if it is a single continuous vein of small width, drifts along the ore on two levels generally will define the orebody sufficiently for stoping purposes. These exploratory drifts generally are a part of the necessary development work in any case, being used for haulage purposes after stoping begins. Raises in the ore from level to level serve similarly, first to explore the orebody, and afterward to provide ventilation and serve as waste passes. It is unusual, however, for such simple means to satisfy all exploration demands. If orebodies are of the massive type or consist of irregular or scattered lenses or even fairly wide veins, the scheme of prospecting and exploration may be complicated greatly.

At Matahambre the orebodies consist of scattered lenses along a more or less continuous belt or zone. These lenses are projected down from upper levels with fair accuracy, being usually regular in dip. On the lower level a main haulage drift is run parallel to the ore zone, and from it crosscuts are driven to intersect the orebodies at their projected positions. If the orebodies are found, they are explored thoroughly on the level by further drifting and crosscutting; if not found where they are expected, the long-hole drill is used to discover their position.

At the United Verde mine the orebodies in what is known as "the main ore zone" can be projected down from level to level with enough accuracy to enable a main haulage drift to be driven just outside the ore as the first step in opening a new level. From this drift enough diamond drilling is done to outline both foot and hanging walls of the orebody and to permit subsequent development work and stoping to be planned to the best advantage.

The use of the long-hole drill for exploring orebodies is gaining rapidly in favor.[6] It has been used extensively at Cananea, for example, taking the place of crosscutting in some instances and assisting in the laying out of development work.

If stopes are silled out on the level, this serves to explore the orebody. This is done sometimes at Nacozari if the orebody is known to be of insufficient height to justify leaving a pillar over the drift. It is the invariable practice at the Magma mine. On the other hand, at the Pecos mine the stope is never silled out on the level. This practice, together with the irregularity of the orebodies, makes it difficult to place the haulage drifts always in the best position and sometimes the sill of the stope is advanced ahead of and 20 feet above the haulage drift on the level in order to take advantage of the exploratory value of the silling-out operation.

An interesting change in exploration-development policies has occurred in the Michigan copper mines in recent years which has played an important part in the remarkably large increase in the grade of ore produced.[7] In these mines the extraction drift on a new level also serves the additional and essential purpose of exploring the lode. Formerly such drifts were driven under contracts which paid for the footage driven regardless of the location of the drift in the lode. The miners often found it advantageous to run the drift next to the footwall slip and consequently the ore from this work was unnecessarily lean. Now the miners have orders to "follow the copper." If they fail, the advance is halted while test holes

6 - Knaebel, J. B., Sampling and Exploration by Means of Hammer Drills: Inf. Circ. 6594, Bureau of Mines, 1932, 29 pp.

7 - Mendelsohn, Albert, Stoping Methods in Michigan Copper Lodes: Min. Cong. Jour., vol. 14, July, 1928, pp. 494-495.

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are drilled or short crosscuts run to the walls to prove the ground, and the miners' contract payments suffer accordingly.

As a stope is carried upward, further exploration is often necessary to insure that all accessible ore of minable grade is being extracted. This usually is done by long-hole drilling. At Matahambre, if a stope narrows suddenly with indications that this may be due to a waste intrusion or to having followed false walls, long holes are drilled from the stope, using either standard drifting machines or a special heavy drill designed for this purpose. For a similar purpose short "stope drifts" or diamond drill holes are used at the United Verde mine to test the genuineness of doubtful walls. At this mine a compact diamond drill weighing only 500 pounds and capable of drilling to a depth of 250 feet has been designed for stope work.

DEVELOPMENT

Development Ratios

Development work has been subdivided in a previous paper of this series into two kinds: Preliminary development and stope development.[8] It is believed that preliminary development work--by which is meant shafts, winzes, adits, drifts, and crosscuts, whether in ore or waste--is relatively independent of stoping methods; whereas stope development--by which is meant the raises and sublevels by which a block of ore is opened for stoping--is an integral part of the stoping method. Comparisons between stope development ratios for various stoping methods should have, therefore, some bearing on the relative efficiencies and costs of those methods. It is useless to attempt to base such comparisons on the "development" footages published by different companies, because there is no uniform, accepted definition of the term, and frequently a company will change its method of classification from time to time. Generally a large part of the stope development footage, as defined here, is classified as stoping. Neither are any figures available showing comparisons of development costs, on this basis, between different stoping methods in a single property. For this reason the theoretical development footages shown in Table 2 have been calculated for several mines, according to such information as is available regarding the orebodies and the mining methods. This tabulation and the accompanying illustrations, Figure 1,A to 1,P, are intended to represent typical but not necessarily average conditions, and it is not implied that the development ratios thus obtained are very close to those obtained in practice. The average development ratio thus calculated is 101 tons per foot of development work, or, eliminating the last three mines, which are working very narrow veins, 121 tons per foot.

Preliminary Development Ratios

The estimates of development ratios given in Table 2 include under the head "Preliminary development" only such drifts and crosscuts as are directly related to the stoping and which are therefore in proportion to the length of the vein; shafts, stations, ventilation raises, and main crosscuts are not considered. The ratio ranges from about 0.001 to 0.024 feet of development work per ton, or from 42 to 1,000 tons per foot of horizontal development along the vein. It bears a rough relation to the thickness of the vein--the wider the vein, the lower the ratio. When the width of the vein is more than 25 feet, however, this relation is obscured by other factors. If, for example, the stope becomes wider than 30 feet the shoveling of broken ore to the chutes may be so costly as to justify an additional extraction drift

8 Jackson, C. F., Shrinkage Stoping: Inf. Circ. 6293, Bureau of Mines, 1930. p. 33.

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Table 2.—Ideal development ratios of various types of cut-and-fill stoping methods

Mine	Figure	Horizontal or Rill stope	Ore blocked out: Width, feet	Length, feet	Lift, feet	Volume, cu. ft.	Tonnage factor, cu. ft. per ton	Tons	Development footage [1]: Preliminary (drifts and crosscuts), feet	Stope (raises), feet	Total, feet	Development ratios: Preliminary development, feet per ton	Stope development, feet per ton	Total development: Feet per ton	Total development: Tons per foot
Cananea	1, A	Horizontal	30	40	125	150,000	10	15,000	A, 30	B, 125	155	0.0020	0.0083	0.0103	97
Cananea	1, B	Rill	50	160	125	1,000,000	10	100,000	A, 160	B, 125	285	.0016	.0012	.0028	351
Pilares	1, C	Horizontal	30	30	100	90,000	12	7,500	A, 30	B, 100	130	.0040	.0133	.0173	58
Pilares	1, D	Rill	30	80	133	320,000	12	26,700	A, 30; A', 30	B, 20; B', 20; C, 113	213	.0023	.0057	.0080	125
United Verde	1, E	Horizontal	50	120	150	900,000	9	100,000	A, 120; B, 120	C, 300; D, 200	740	.0024	.0050	.0074	135
Campbell	1, F	Rill	50	60	100	300,000	9	33,300	A, 50; B, 120.	C, 100; D, 100	370	.0051	.0060	.0111	90
Magma	1, G	Rill	16	50	200	160,000	10.5	15,250	A, 16; B, 16; C, 50; E, 32	D, 200	314	.0075	.0131	.0206	49
Matahambre	1, H	Horizontal	40	100	150	600,000	10.5	57,150	A, 100; B, 100	C, 28; D, 185	413	.0035	.0037	.0072	138
Pecos	1, I	Horizontal	15	25	100	37,500	10.5 [2]	3,570	A, 15	B, 20; C, 20	55	.0042	.0112	.0154	65
Arizona Copper	1, J	Rill	25	150	150	562,500	10	56,250	A, 150	B, 150	300	.0027	.0027	.0055	187
Copper Range	1, K	Horizontal	18	400	100	720,000	10.8	66,700	A, 400	B, 100	500	.0060	.0015	.0075	133
Champion	1, L	Rill	17	200	100	340,000	11	30,900	A, 200	B, 100; C, 400 [3]	700	.0065	.0162	.0227	44
United Eastern	1, M	Horizontal	18	100	150	270,000	12 [2]	22,500	A, 100	B, 150	250	.0044	.0067	.0111	90
Iron Cap	1, N	Rill	5	125	100	62,500	11	5,680	A, 125	B, 100	225	.0220	.0176	.0396	25
Eighty-Five	1, O	Rill	5	120	100	60,000	12 [2]	5,000	A, 120	B, 100; C, 26; D, 52; E, 90 [3]	388	.0240	.0536	.0776	13
Zaruma	1, P	Horizontal	5	100	98.4	49,200	12	4,100	A, 100	B, 98; C, 200	398	.0244	.0726	.0970	10
Averages								34,400	138 (249 tons per foot)	201 (171 tons per foot)	340	.0040	.0058	.0099	101
Averages, excluding last three								41,400	144 (287 tons per foot)	197 (210 tons per foot)	341	.0035	.0048	.0082	121
Averages, last three only								4,930	115 (43 tons per foot)	222 (22 tons per foot)	337	.0233	.0451	.0683	15

[1] Letters refer to letters on diagrams.

[2] Assumed.

[3] A subdrift, properly stope development.

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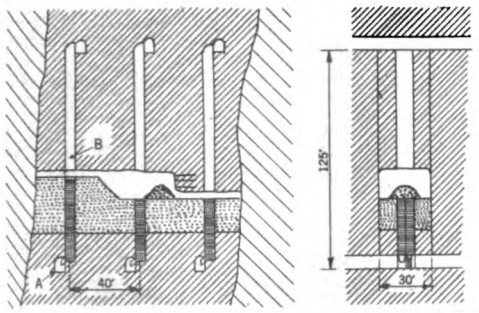
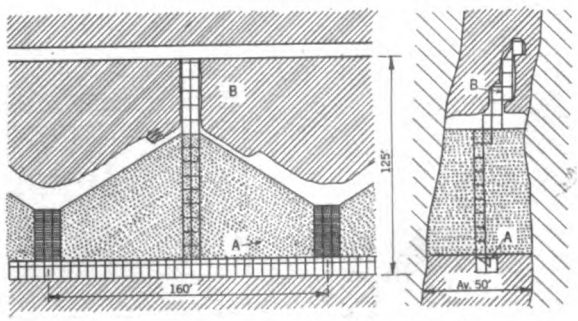


Area, 30 by 40 ft. = 1,200 sq. ft. (per raise)
Lift, 125 ft.
Tonnage, 150,000 ÷ 10 = 15,000 tons

	Feet	Feet per ton
Drift A	30 =	0.0020
Raise B	125 =	.0083
Total	155 =	.0103, or 97 tons per ft.

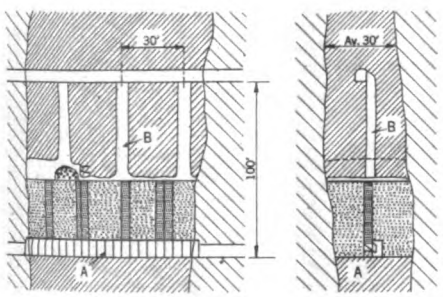
A, CANANEA: HORIZONTAL CUT-AND-FILL STOPE-AND-PILLAR METHOD



Area, 50 by 60 ft. = 8,000 sq. ft.
Lift, 125 ft.
Tonnage, 1,000,000 ÷ 10 = 100,000 tons

	Feet	Feet per ton
Drift A	160 =	0.0016
Raise B	125 =	.0012
Total	285 =	.0028, or 351 tons per ft.

B, CANANEA: RILL CUT-AND-FILL STOPE

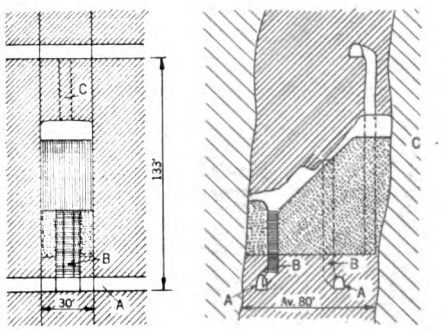


NOTE: stopes are sometimes silled 20 feet above level

Area, 30 by 30 ft. = 900 sq. ft (per raise)
Lift, 100 ft.
Tonnage, 90,000 ÷ 12 = 7,500 tons

	Feet	Feet per ton
Timbered gangway A	30 =	0.0040
Fill raise B	100 =	.0133
Total	130 =	.0173, or 58 tons per ft.

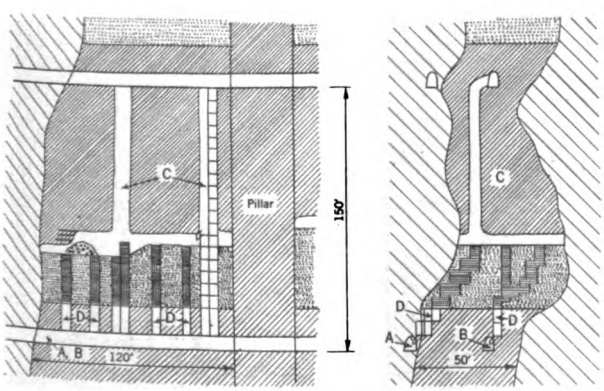
C, PILARES: HORIZONTAL CUT-AND-FILL STOPE



Area, 30 by 80 ft. = 2,400 sq. ft.
Lift, 133 ft.
Tonnage, 320,000 ÷ 12 = 26,700 tons

	Feet	
Drift A	30	
(possibly) Drift A'	30	Feet per ton
Total drift	30 or 60 =	0.0011 or 0.0023
Raise B	20	
(possibly) Raise B'	20	
Raise C	113	
Total raise	133 or 153 =	.0050 or .0057
Total drift and raise	163 or 213 =	.0061 or .0080, or 164 or 125 tons per ft.

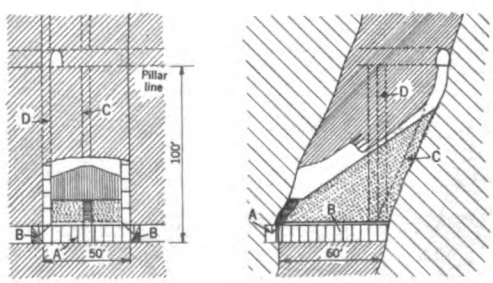
D, PILARES: RILL CUT-AND-FILL STOPE



Area, 50 by 120 ft. = 6,000 sq. ft
Lift, 150 ft.
Tonnage, 900,000 ÷ 9 = 100,000 tons

	Feet	
Drift A, on contact	120	
Drift B, in ore	120	Feet per ton
Total drift	240 =	0.0024
Fill raises C (2)	300	
Chute raises D (8)	200	
Total raises	500 =	.0050
Total drift and raise	740 =	.0074, or 135 tons per ft.

E, UNITED VERDE: HORIZONTAL CUT AND FILL



Area, 50 by 60 ft. = 3,000 sq. ft
Lift, 100 ft.
Tonnage, 300,000 ÷ 9 = 33,300 tons

	Feet	
Drift A	50	
Crosscuts B (2)	120	Feet per ton
Total drift	170 =	0.0051
Fill raise C	100	
Manway raise D	100	
Total raise	200 =	.0060
Total drift and raise	370 =	.0111 or 90 tons per ft.

F, CAMPBELL: RILL CUT-AND-FILL STOPE

Figure 1.- Diagrammatic sketches of ideal cut-and-fill stopes

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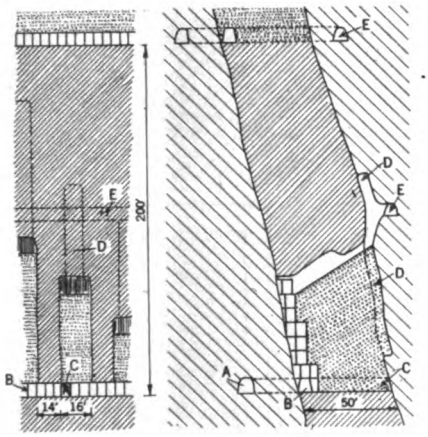
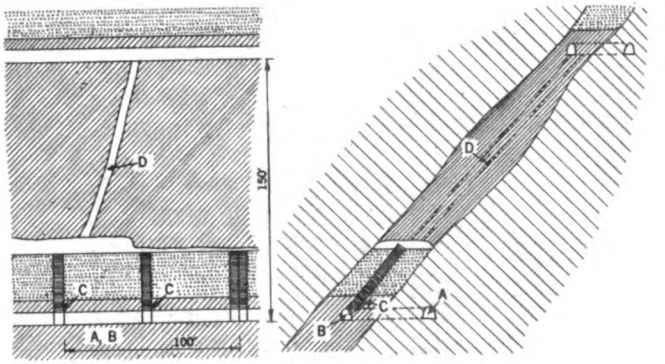


Area, 16 by 50 ft. = 800 sq. ft.
Lift, 200 ft.
Tonnage, 160,000 ÷ 10.5 = 15,250 tons

	Feet	
Haulage drift A	16	
Extraction drift B	16	
Crosscut C	50	
Fill drifts E (2)	32	Feet per ton
Total drift	114	= 0.0075
Fill raise D	200	= .0131
Total drift and raise	314	= .0206, or 49 tons per ft.

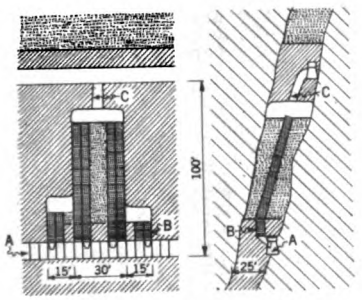
G, MAGMA: RILL CUT-AND-FILL STOPE-AND-PILLAR METHOD



Area 40 by 100 ft. = 4,000 sq. ft.
Lift, 150 ft.
Tonnage, 600,000 ÷ 10.5 = 57,150 tons

	Feet	
Haulage drift A	100	
Extraction drift B	100	Feet per ton
Total drift	200	= 0.0035
Chute raises C (2)	28	
Fill raise D	185	
Total raise	213	= .0037
Total drift and raise	413	= .0072, or 139 tons per ft.

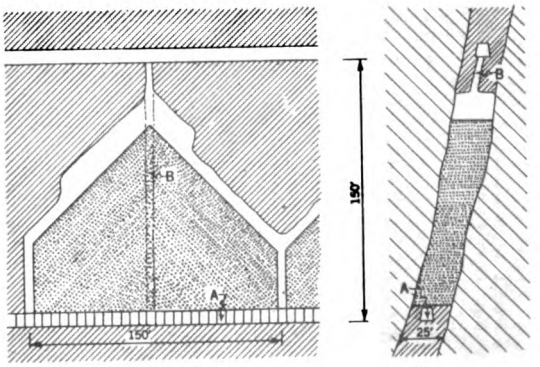
H, MATAHAMBRE: HORIZONTAL CUT-AND-FILL STOPE



Area, 15 by 25 ft. = 375 sq. ft.
Lift, 100 ft.
Tonnage, 37,500 ÷ 10.5 (assumed) = 3,570 tons

	Feet	
Drift A	15	
Raise B	20	
Fill raise	20	(assuming one new fill raise 100 ft. long for every five stope sections)
Total	55	= 0.0154 ft. per ton = 65 tons per ft.

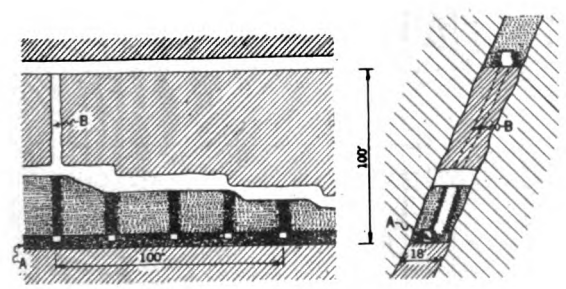
I, PECOS: HORIZONTAL CUT AND FILL STOPES IN NARROW SECTIONS WITHOUT PILLARS



Area, 25 by 150 ft. = 3,750 sq. ft.
Lift, 150 ft.
Tonnage, 562,500 ÷ 10 = 56,250 tons

	Feet	
Drift A	150	
Raise B	150	
Total	300	= 0.0053 ft. per ton = 187 tons per ft.

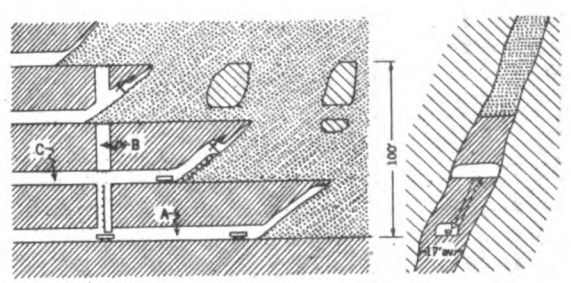
J, ARIZONA COPPER CO.: FILLED RILL STOPE



Area, 18 by 400 ft. = 7,200 sq. ft. (Fill raises 400-500 ft. apart)
Lift, 100 ft.
Tonnage, 720,000 ÷ 10.8 = 66,700 tons

	Feet	
Drift A	400	(driven full width of ore)
Raise B	100	
Total	500	= 0.0075 ft. per ton = 133 tons per ft.

K, COPPER RANGE CO.: HORIZONTAL CUT-AND-FILL STOPE



Area, 17 by 200 ft. = 3,400 sq. ft.
Lift, 100 ft.
Tons broken, 340,000 ÷ 11 = 30,900
Tons of ore produced, 30,900 less 40% = 18,500

	Feet	
Drift A	200	
Raise B	100	
Subdrifts C	400	(driven full width of ore)
Total	700	= 0.0226 ft. per ton broken = 44 tons per ft. = 26 tons of ore (hoisted) per ft.

L, CHAMPION: FILLED RILL

Figure 1.- Diagrammatic sketches of ideal cut-and-fill stopes - Continued

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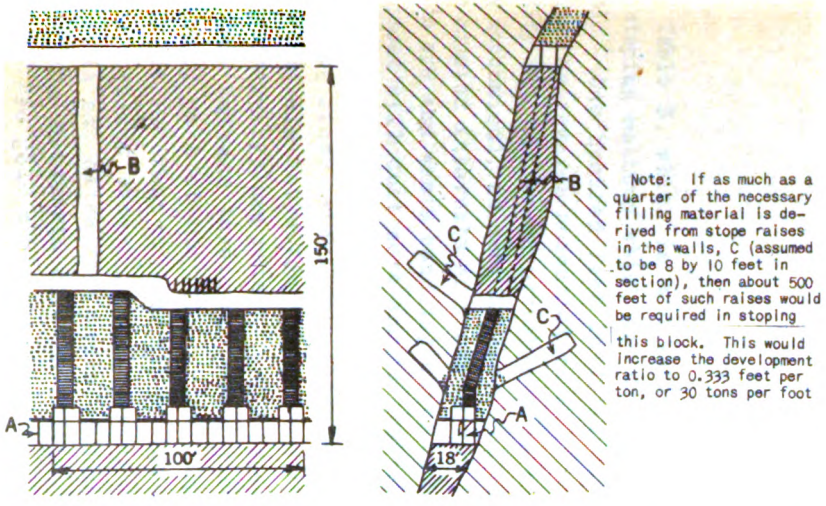
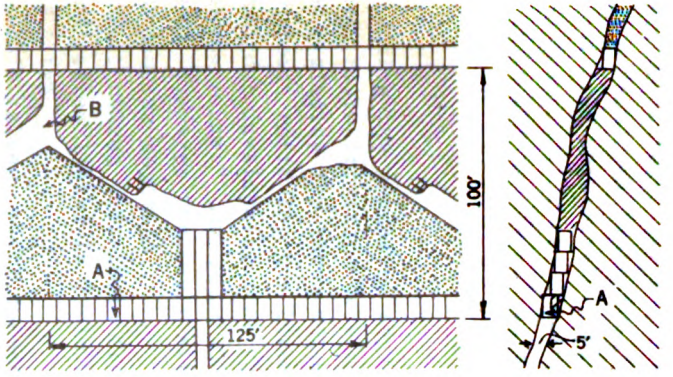


Note: If as much as a quarter of the necessary filling material is derived from stope raises in the walls, C (assumed to be 8 by 10 feet in section), then about 500 feet of such raises would be required in stoping this block. This would increase the development ratio to 0.333 feet per ton, or 30 tons per foot

Area, 18 by 100 ft. = 1,800 sq. ft.
Lift, 150 ft.
Tonnage, 270,000 ÷ 12 = 22,500 tons

		Feet
Drift A		100 (assumed interval between fill raises
Fill raise B		150
Total		250 = 0.0111 ft. per ton
		= 90 tons per ft.

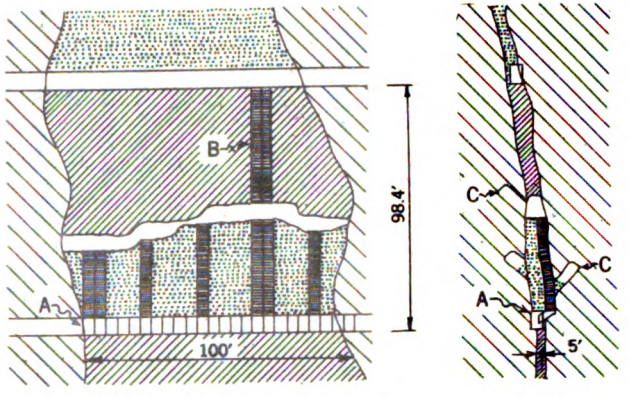
M, UNITED EASTERN: HORIZONTAL CUT-AND-FILL STOPE



Area, 5 by 125 ft. = 625 sq. ft.
Lift, 100 ft.
Tonnage, 62,500 ÷ 11 = 5,680 tons

	Feet
Drift A	125
Raise B	100
Total	225 = 0.0396 ft. per ton
	= 25 tons per ft.

N, IRON CAP: FILLED-RILL STOPE

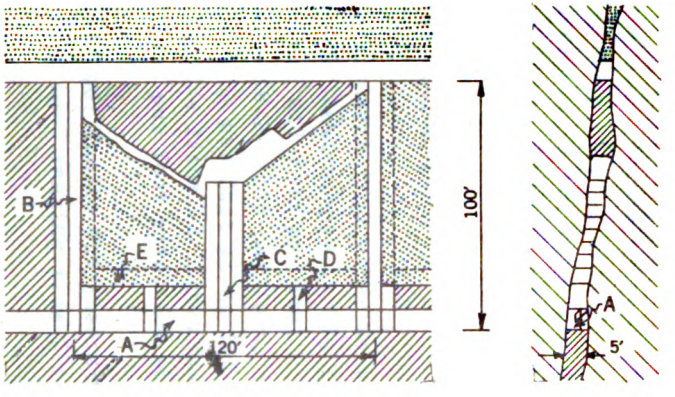


Area, 5 by 100 ft. = 500 sq. ft.
Lift, 98.4 ft. (30 meters)
Tonnage, 49,200 ÷ 12 = 4,100 tons

		Feet
Drift A		100
Ventilation raise B		98
Stope raises C		200 (200 ft. of 5 by 6 ft. raise would supply
Total		398 approximately a quarter of the necessary
		filling material)

Development ration = 0.097 ft. per ton
= 10 tons per ft.

P, ZARUNA, ECUADOR: HORIZONTAL CUT-AND-FILL STOPE



Area, 5 by 120 ft. = 600 sq. ft.
Lift, 100 ft.
Tonnage, 60,000 ÷ 12 = 5,000 tons

	Feet
Drift A	120
Raise B	100
Raise C	26
Raises D (2)	52
Subdrifts E (2)	90
Total	388 = 0.0776 ft. per ton
	= 13 tons per ft.

O, EIGHTY-FIVE: FILLED-RILL STOPE

Figure 1.- Diagrammatic sketches of ideal cut-and-fill stopes - Continued

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and an additional line of chutes. Moreover, ground conditions in the vein, or congested haulage conditions, may require a main drift in the country rock. Thus, it is apparent that a high development ratio does not necessarily condemn a stoping method; it may be rather an expression of unfavorable mining conditions. Conversely, a low ratio may be the result of a costly and shortsighted stoping method rather than of an efficient one. Obviously, development costs can not be judged separately from stoping costs.

Stope Development Ratios

Making the suggested segregation of stope development from other development work, the average ratio for stope development of this group of ideal stopes is 171 tons per foot, or, again excluding the three mines working narrow veins, 210 tons per foot. The average stope development ratio for the three narrow vein deposits is 22 tons per foot, which represents a much higher development cost than the figure of 100 tons per foot given for shrinkage stoping in narrow veins.[9] The ratio of 171 tons per foot for the first 13 listed cut-and-fill stopes, however, compares favorably with the ratio of 163 tons per foot given for shrinkage stopes in wide, tabular deposits.

Fill raises contribute from 0.001 to 0.020 foot per ton to the theoretical stope development ratios listed in Table 2, which constitutes a cost ranging from 1 cent to perhaps 30 cents a ton.

Chute raises, where necessary, account for development footages ranging from as little as 0.0005 to as much as 0.015 foot per ton of ore.

Preliminary Development

Level Intervals

Table 3, which is a compilation for comparative purposes of data regarding the mines and stoping methods which are discussed in this paper, shows the vertical intervals between levels. One hundred feet is decidedly the commonest interval. This is customary and convenient as regards nomenclature and visualizing the workings; however, it has little else to justify it. It seems probable from a consideration of the list that a larger interval is more economical, as many of the older mines which have had time and opportunity to experiment have adopted intervals as great as 200 feet. Marked economies in main level development costs and in ore and waste handling costs can be obtained by increasing the interval. Vertical development will cost more and is sometimes the factor which limits the level interval. This added cost, however, may be partly avoided by using sublevels, as, for example, at Magma and Ground Hog mines, where only alternate levels are fully developed for haulage and the intermediate levels serve for travel, ventilation, supplies, and to decrease the cost of driving the excessively long raises. Some types of ore are so destructive to chutes that the greater intervals between levels are impracticable, because of excessive chute repair costs during the latter part of the life of the stopes. It is suggested also that placing the level intervals at 100 feet or less may be the better policy for a shallow, young mine, particularly if the downward extent of the orebody is not known or reasonably certain.

It is not often that mines of considerable size can be worked through adits or tunnels alone. The Britannia mine in British Columbia is one in which almost no ore or waste is hoisted. The gold mine of the Syndicate Mining Co., Philippine Islands, is worked to a depth of 500 feet through two adits spaced 250 feet apart. The Coronado mine of the Arizona

9 - Ibid, p. 36.

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Copper Co. was worked chiefly or entirely through adits. At all the other mines listed in this paper ore and waste are hoisted through shafts. However, adits have been used in the development of about two-thirds of them, and more than one-third have their main ore hoists underground. This would appear to have no connection with the use of the cut-and-fill stoping method. It is possible, however, that the general freedom from subsidence when using a filled-stope method permits the use of underground installations and a closer approach to the orebody by this means, as well as the saving of hoisting costs.

Main adits are driven generally with larger cross sections than drifts, because of the greater amount of traffic through them. Examples taken from the mines included in this paper range in size from the 7 by 7 foot Stillwell tunnel in the Telluride district to the 10 by 12 foot main-haulage adits of the United Verde and Pilares mines. Grades of adits are usually ¼ to ½ of 1 per cent in favor of the loaded trains, which at the same time facilitates mine drainage.

Shafts

Less than one-quarter of the shafts at the mines considered in this paper are inclined; these are chiefly mines of a relatively low daily tonnage. The Copper Range mines and the Bunker Hill and Sullivan mines are exceptions to the latter statement. In the former mine the great depth of the lodes, dipping at 70°, and in the latter the flat dip of the vein combined with considerable depth, make vertical shafts uneconomical. Vertical shafts are generally preferable to inclined ones.

The location of mine shafts is an important phase of the development of all deep mines, but one which is often given insufficient consideration. The method of mining enters into this problem insofar as it affects ground movement or subsidence. Knowledge of the mechanics of ground movement is inadequate to permit a confident prediction of what will happen in any given case; this is especially true at a new mine where the characteristics of the ore and country rock are not well understood. If a caving system of mining is used, such as top-slicing, subsidence will commonly take place on as flat an angle as 40° to 45° away from the lowest point of caving. Crane[10] has recorded many instances in the copper and iron mines of Michigan where movement has occurred on angles as low as 35°. Careful filling of stopes will usually prevent the initial movement which leads to subsidence. Although some settling is likely to occur where large areas are undercut, proper filling will restrict this movement to a minimum. Therefore, the use of a cut-and-fill method will sometimes permit the location of shafts and surface plant close to the orebody with resulting lower costs.

Other phases of shaft design, such as size, shape, and material of construction, are not closely related to stoping methods. The cut-and-fill method of mining theoretically requires a minimum of timbering. Actually most mines using this method find it necessary to use considerable amounts of timber, not only for drifts and raises, but also for stoping. Therefore shafts at such properties are called upon to handle timber as well as less bulky supplies. Three of the mines considered in this paper, the Campbell, the Pilares, and the United Verde, have shafts containing a large service-cage compartment. Evidently only a large daily tonnage justifies the expense of such an installation. Almost all of the large porphyry mines have such shafts.

Almost all ore hoisting is now done with skips rather than with cages, as was the practice in former times; skips, of course, are necessarily used in inclined shafts. The Ground Hog is the only mine in the group discussed here that uses cage hoisting exclusively. At

10 Crane, W. R., Subsidence and Ground Movement in the Copper and Iron Mines of the Upper Peninsula, Michigan: Bull. 295, Bureau of Mines, 1929., 66 pp.

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this mine, cage hoisting is thought to be almost essential because of the waste and number of classes of ore handled. Of course, almost all vertical-shaft mines handle some or all development waste and some ore from new levels by means of cages, to avoid the immediate construction of skip-loading pockets or to avoid interfering with ore hoisting. La Colorada mine, Cananea, during the development of new levels, hoists as much as 500 tons of ore and waste per day by means of two triple-deck cages. Handling waste by means of cages is particularly suitable to the cut-and-fill method of mining because the waste generally is needed for filling stopes and, if skips were to be used, underground skip dumps and waste pockets would be required.

Drifts and Crosscuts

Usually crosscuts must be driven from the shaft to the vein or orebody. At some mines the shafts are located so fortunately that these crosscuts are little longer than the requirement for a good shaft station. In such cases they are driven full station width until they intersect the vein, where drifts are turned off in either direction. Other conditions generally prevail, however, and in most mines crosscuts over 100 feet long are required to reach the ore. At the United Verde mine, in order to secure minimum average tramming distance, the main crosscut on each level is directed toward the center of the orebody as projected downward from the last level. At Cananea, where the shaft is about 300 feet north of the large, elliptical orebody, a service crosscut is driven south from the shaft and through the orebody, the development laterals being then turned off from it in both directions parallel to the long axis of the ellipse. In addition, on the alternate levels which are to be haulage levels, a main haulage drift is run close to and back of the shaft and continued in a complete loop around the orebody, intersecting the ends of the laterals. A haulage loop around the shaft is not always justifiable, as it involves considerable extra footage. At the Magma mine the haulage drifts are extended over the shaft pocket and past the shaft itself far enough to form a tail drift. Where the ore is hoisted in cages no tail room is necessary, but the shaft station must be laid out to provide switching room for loaded and empty cars.

Where the hoisting shaft is rather flatly inclined, the station almost necessarily extends over and parallel to the shaft far enough to give vertical clearance for a loading pocket. This is illustrated by the practice at the Bunker Hill and Sullivan mine. At the Copper Range Co. mines the shafts are in or close to the lode. If the shaft is in the lode, the ore is dumped directly into the skips from the cars. If the shaft is in the footwall, loading pockets may be provided.

Crosscuts from the shaft to the orebody may be driven larger than ordinary drifts, to permit free passage of men and trains, or even to allow double tracking; usually, however, they are the same size as standard drifts.

Drifts in the cut-and-fill mines considered here are of every variety. Apparently the location, size, or method of support of drifts bear little or no relation to the general stoping method used, whether cut-and-fill, shrinkage, square-set, or other method, but depend rather on the position and characteristics of the orebodies and country rock, and on the scale of operations.

The first essential preliminary development opening, after the shaft has been sunk and a crosscut driven to the vein, is an extraction drift. In vertical or steep-dipping orebodies the most convenient location for this drift is in the ore midway between the walls. If the ore dips more flatly and is not more than about 15 feet wide, the best location for the extraction drift is at the footwall. This permits the most efficient placing of ore chutes under the stope. In a wide orebody more than one extraction drift may be required, as at

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Cananea (see fig. 2) where the extraction drifts or laterals are driven lengthwise through the orebody at 40-foot intervals. As the stopes and pillars are laid out at right angles to these drifts, extraction chutes are spaced at 40-foot intervals along the center-lines of the stopes, giving a maximum shoveling distance in the stopes of 25 to 30 feet. At Pilares a single drift in the center of an orebody is sometimes sufficient for extraction purposes. At Matahambre a drift approximately in the center of the vein is driven for extraction. However, many different factors may force the location of an extraction drift elsewhere than in the center of the ore. Probably the most common of these is the character of the ground. Very often a drift close to one wall of an orebody either in the ore or in the country rock can be maintained much more cheaply than one in the center of the vein. At the Ground Hog mine the ore is relatively heavy and weak and the hanging wall is stronger than the footwall, the latter being chiefly a thick soft seam of gouge. Therefore, the extraction drift is run in the ore along the hanging wall. At the Pecos mine the ore and walls are both weak; the extraction drift is run in the ore close to the footwall.

When inclined cut-and-fill stopes are laid out at right angles to the strike in wide, steeply inclined veins, it would seem that the rill ought to dip toward the footwall to give the best support to the walls and to expose the least amount of back at one time. This would place the extraction drift on the footwall. However, no uniformity is found in this respect.

Frequently the extraction drift, due to its closeness to stoping operations, is difficult to maintain and must be supplemented for haulage purposes by another drift. Moreover, if more than two or three stopes are working along a vein the use of a single drift for tramming from all of them causes great congestion. Therefore, as, for example, at the Broken Hill, Magma, Matahambre, Park Utah, and Zaruma mines, main haulage drifts are run parallel to the vein and some distance away from it, either in the foot or hanging wall. At the United Verde mine the sulphide ore is so hard, and drifting in it is therefore so expensive, that extraction drifts are driven in country rock close to the edge of the massive sulphide orebody. If the entire width of the stope can not be reached economically from a drift in this position, then drifts in ore become necessary in spite of their high cost.

Drifts in the hanging wall are useless for extraction purposes except in steeply dipping veins. Magma uses an extraction drift in the ore on the footwall, a haulage drift 25 feet away from the ore in the footwall, and a waste drift in the hanging wall close to the ore for stope-filling purposes. In this mine, as at Matahambre and Broken Hill, the main haulage drift is run first and then crosscuts driven from it through the orebody. At the Campbell mine of the Calumet & Arizona Mining Co. the main haulage drift is driven in the country rock of the hanging wall close to the ore, and extraction crosscuts driven from it through the ore body on the lines between stope and pillar sections. Contrary to the general practice, no drift is run in the ore at this mine.

At Pilares the main motor-haulage drifts are 6 by 8 feet in section, and other drifts and crosscuts 5 by 7 feet. At the Pecos mine, similarly, the short crosscuts from the shaft to the veins are 7 feet high by 8 feet wide; drifts in the ore are timbered 7 feet high in the clear, 5 feet 4 inches wide at the top, and 5 feet 10 inches wide at the bottom. Numerous examples could be cited without establishing any fixed rules for the size and method of support of these openings. The commonest cross section is 5 by 7 feet for drifts, with crosscuts usually of the same size though sometimes larger. It is believed that it is good practice to make the more traveled drifts and crosscuts, such as those between the shaft and orebody, larger than this, if only from the standpoint of safety. In heavy ground small drifts crush less readily than large ones, but if some crushing is inevitable before the useful life of the opening is finished, allowance should be made for this and the section chosen should be large enough to permit passage of cars without frequent chopping out of posts or caps. Most of the mines considered here use little or no timber in horizontal de-

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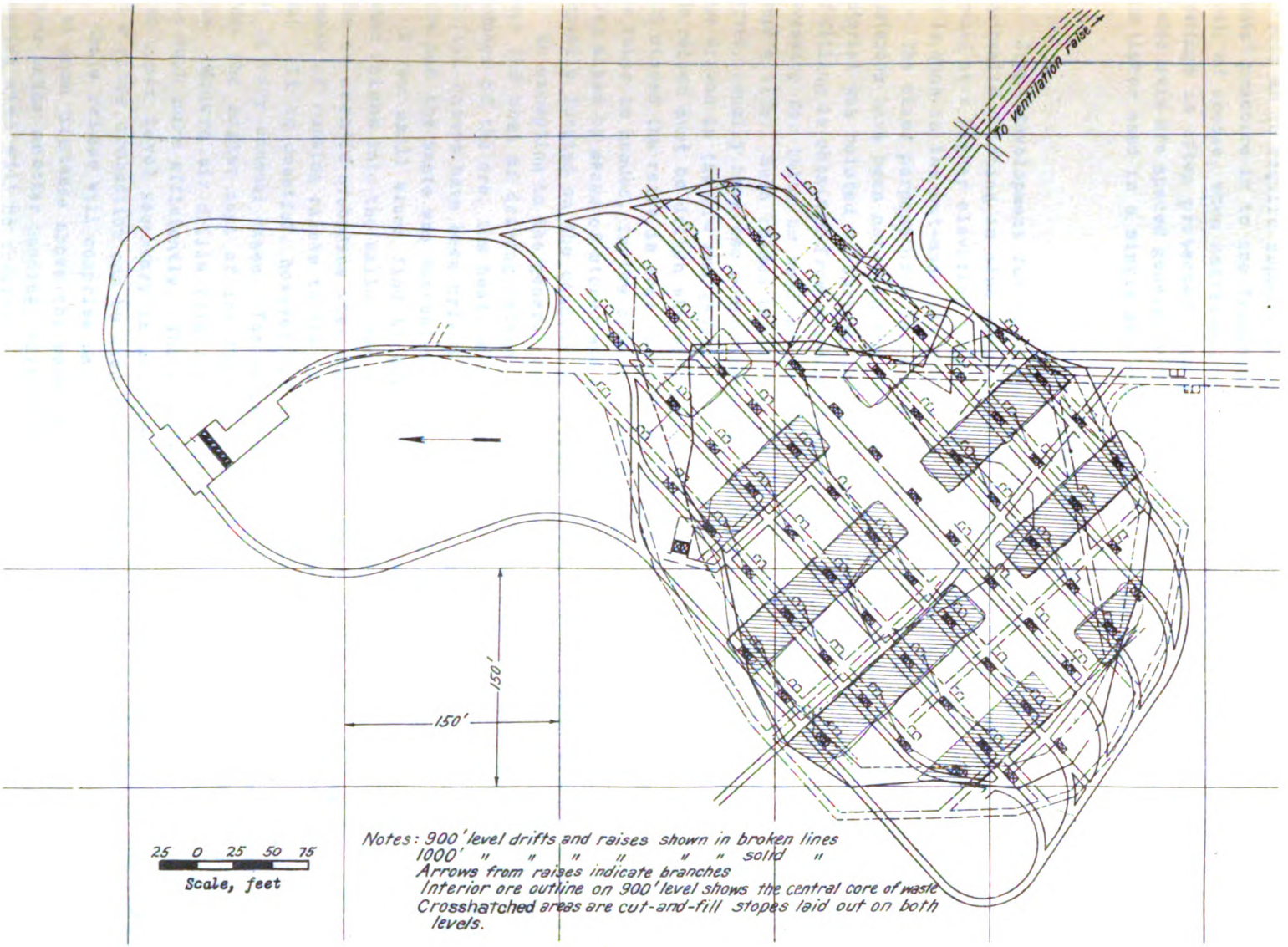
To ventilation raise
150'
150'
25 0 25 50 75
Scale, feet
Notes: 900' level drifts and raises shown in broken lines
1000' " " " " " " solid "
Arrows from raises indicate branches
Interior ore outline on 900' level shows the central core of waste
Crosshatched areas are cut-and-fill stopes laid out on both levels.

Figure 2.- Typical haulage and development levels, La Colorada mine, Cananea

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velopment openings, at least previous to the start of mining operations overhead. This is probably a reflection of the fact that cut-and-fill mining would not be well adapted to ground so soft as to require support in such small openings. When timbering is employed, the commonest practice is to use framed Oregon fir. Other varieties of timber and round timber are used, of course, when satisfactory quality is available at lower cost. Timber in permanent openings is often protected with preservatives such as fluorides, chlorides, or creosote. Timber sets are spaced generally 5 feet apart, though this spacing ranges from 3½ to 5½ feet. The timber used in a single set ranges from 150 to 300 board feet.

Stope Development

Stope development for cut-and-fill mining usually involves only raising, although occasionally drifting is also necessary. The necessity for drifting occurs when the stope is opened at a higher elevation than the level by running a sublevel drift from raise to raise, as is done in the cut-and-fill stopes at the Eighty-Five mine.

The chief purpose of a raise from level to level is to bring fill into the stope. Rare instances have been noted in the early accounts of cut-and-fill mining in which the filling material was hoisted or carried into the stope from below, but the universal practice now, if filling is obtained from outside the stope, is to drop the fill in from above. Raises are necessary for this, as well as for ventilation, supplies, and travel, at all periods of a stope's life. Such raises usually start at the level, and because of the several purposes served, usually have two compartments as indicated in Figure 1. Fill raises are almost always driven in the ore and generally on the hanging wall. In an inclined cut-and-fill stope the raises must be driven where the high end of the rill is to be. In horizontal cut-and-fill stopes the raise is most conveniently placed in the center of the stope or, if more than one raise is needed, in the center of roughly equal areas. However, when a block of ore is being mined by means of stopes and pillars, or by a panel system, the raises may be most conveniently located on the boundaries between stopes, so that each may serve two areas.

An exception to the general rule of driving the fill raises in ore was at the Magma mine. There the cost of driving raises from level to level in the ore was so high, owing to the hardness of the ore, the heat, and the great distance between levels, that for several years all fill raises have been driven in the hanging wall. This resulted in much lower raising costs and the waste was conveniently used as filling in the stope.

A few small mines find it advantageous to obtain the fill for stopes by driving inclined raises into the walls (see figs. 1,M and 1,P). This practice is economical in narrow veins or erratic orebodies when the amount of waste needed for filling does not justify the expense of running raises to the level above or where waste for filling must be mined in any event. It is doubtful, however, if this practice could be justified on a large scale, except in very unusual cases. The undoubted value of the openings as prospects would scarcely offset the higher cost of the fill thus obtained and the possible weakening of the stope walls. Modern air drills with long or sectional drill steel can be used to prospect the walls much more efficiently. The requirements for ventilation and safety make a connection to an upper level necessary in most cases before stoping reaches any considerable height; such a raise connection can be used for bringing in fill.

Chute raises will comprise part of the stope development footage if the stope is silled out at some distance above the back of the drift. Their spacing, which will be discussed further under another heading, varies from about 15 to as much as 40 feet. Chutes and raise timbering also will be treated under later headings.

A recent innovation in stoping methods at the Champion mine of the Copper Range Co., Michigan, is shown diagrammatically in Figure 1,L, which should be compared with Figure 1,K, illustrating the older stoping scheme. The block of ore between levels is split into three 25-foot lifts by two sublevels driven from the raise. These sublevels are driven 8 feet

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high and the full width of the ore, and the muck is scraped to the raise. The sublevels serve a very important purpose in exploring the orebody and these, as stoping progresses, become the sills of the rill stopes. The effect is to create three stope sections where formerly there was one. The new method involves a great increase in stope development footage, but offers advantages which outweigh the increased development cost.

STOPING

In planning a cut-and-fill stoping campaign, a choice must be made first between horizontal and rill stoping. This done, the laycut of the stopes and the size of stope sections must be decided on. In a new operation these decisions may not be final until much of the development work described in the preceding pages is complete.

The first operation in cut-and-fill stoping is silling out, which consists of removing a horizontal slice of ore over the entire area of the block of ore to be mined as a stope. If it is done on the level, silling out is followed by timbering an extraction drift and building chutes and manways at the desired intervals along the drift. If the stope is silled at some distance above the level, extraction chutes and raises usually are made ready in advance, so that as the sill face advances it will encounter the tops of the chute raises.

If horizontal cut-and-fill stoping is to be done, the routine cycle of filling, breaking and removing the ore starts at once upon completion of the above steps. If inclined, or rill, cut-and-fill stoping has been decided on, the next step after silling out is to cut successive inclined slices from the back of the stope, starting at a waste fill raise. Usually two or three cuts, depending upon the safety with which work can be done under a high back, are taken before the pile of broken ore is cleaned out and the fill placed. This fill is not spread, but is allowed to remain at its natural angle of repose and generally is covered with a temporary floor of plank. Another cut is then mined up the slope, the broken ore running to the bottom of the slope. Filling and mining alternate and the length of the rill increases until the toe reaches the chute which is to form the end of the stope. After that, the chute must be built up after each cut and before filling.

With either the horizontal or the rill method some change in procedure will be necessary upon reaching the level above. Most commonly square-setting is adopted, as this method is well adapted to heavy ground and to working under old filled stopes.

Horizontal Versus Inclined Cut-and-fill Methods

The idea behind the inclined cut-and-fill stoping method, as behind many similar ones, is to reduce or eliminate hand shoveling. However, the choice between the inclined cut-and-fill stope and the horizontal depends on so many factors that few generalizations can be made. Rill stoping when practicable usually gives lower costs than horizontal stoping. A tabulation of the mines using the cut-and-fill method, however, shows that only slightly more than half of them use the rill method, and the inference is that conditions in the other mines prohibit the use of the lower cost method.

Ground conditions are the most important factor in making the choice between the two methods. If loose or running ground is sometimes encountered in the stope, the result in a rill would be to flood the part of the stope below the run, whereas on a horizontal floor the run would tend to seal itself and stop. If the ore is so loose and blocky as to require much support in the form of stulls or cribbed bulkheads, the rill method is not practical. Such support is difficult to place on the steeply sloping floor, and is always in danger of being knocked out by moving broken ore or fill. A third element of ground conditions which is of prime importance in almost all mining operations is the extent and orientation of

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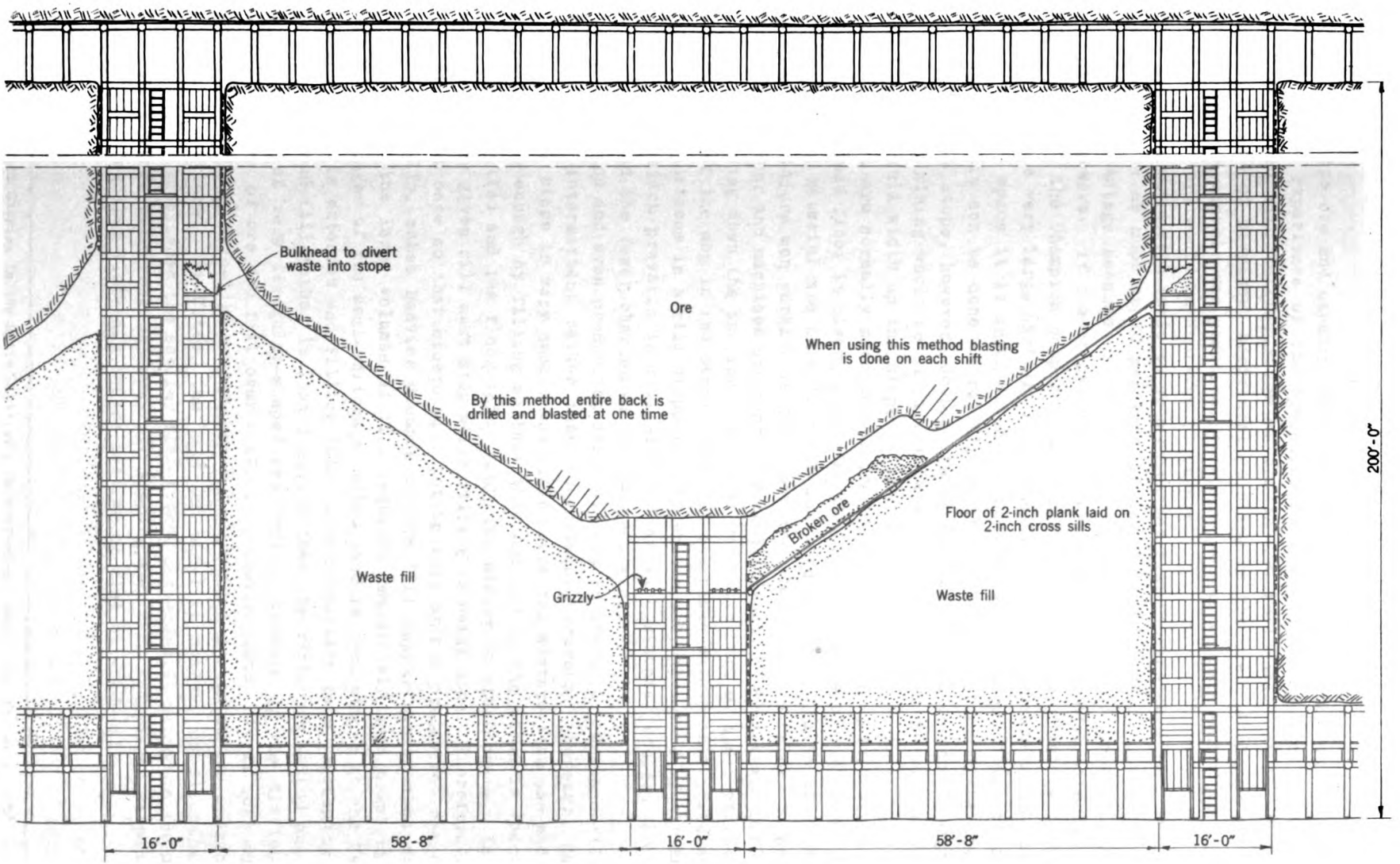

Figure 3.— Rill cut-and-fill stope at Butte

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planes of weakness in the ore and country rock. An excellent illustration of the importance of this factor is the experience of the operators at the Ground Hog mine. Considerable trouble was experienced with falling slabs of ore in the horizontal cut-and-fill stopes because of the presence of prominent horizontal jointing in the ore. A change was made to the rill method and the trouble was practically eliminated, as in this method of stoping no one large flat slab of ore would be deprived of support.

If the ore occurs mixed with waste to such an extent that considerable sorting is necessary in the stopes, say as much as 20 per cent of the material broken, the rill stope is ordinarily at a disadvantage because of its limited storage space. A moderate amount of sorting is possible, however, if the stope is properly laid out, and under exceptional circumstances, such as at the Champion copper mine, Michigan, it may be possible to sort out and leave in the stope a very large part of the ground broken. For reasons likewise related to the lack of storage space it is inconvenient to handle more than one class of ore in a rill stope, whereas this can be done as readily in the horizontal stope as in any mining method. Even in a rill stope, however, double chutes can be provided to handle high or low grade ores. Selective mining would be difficult in a rill stope, because of the necessity of advancing the face full width up the slope of the rill.

Working conditions are normally much better in a horizontal cut-and-fill stope than in a rill stope. An inclined floor is always a poor working place because of the insecure footing. Falls of men and material are more frequent and more serious. Falls of rock in any part of the stope may injure men working farther down the slope. It is harder to move from place to place, and timber and machines are handled with difficulty. Pipe lines and hose are in danger from rock moving down the incline. Men must work with part of their attention focused on all the rest of the men in the stope. Against all these unfavorable items going to make up the working conditions in a rill stope is only one on the other side of the balance; somewhat better ventilation prevails in rill stopes than in horizontal stopes, due to the better natural draft and the fewer chances of short-circuiting.

Continuous operation and even production are desirable in any mine. Cut-and-fill stoping is essentially an intermittent rather than a continuous process. However, unless a horizontal cut-and-fill stope is very small, the mining face can advance from one end to the other, followed closely enough by filling so that when one end of the stope is reached the other will have been filled and the floor laid, ready for mining to start again. In a rill stope, the mining on any given rill must stop while filling is being done. Therefore, enough rills must be worked at once so that miners will not be idle or ore production cease. The rill method, incidentally, makes heavier demands on the fill supply system than does the horizontal, because of the large volumes of fill required immediately upon completion of a cut. One of the advantages of the semi-shrinkage method used in some stopes at the Campbell mine is that the cycle is extended and filling interrupts production less frequently.

The inclined cut-and-fill method is less flexible than the flat-back method and is not adapted to the mining of very irregular-shaped ore bodies, because of the difficulty of maintaining gravity flow of ore and fill over a floor of uneven width. It is most suitable for the working of uniform, vein-like or tabular deposits which are extensive enough to be divided into large sections. At Butte, the rill cut-and-fill method, as illustrated in Figure 3,[11] has largely superseded the earlier flat-backed cut-and-fill method in the narrower veins. In these mines both rill cut-and-fill stoping and rill square-set stoping show higher labor efficiencies than flat-back square-setting or flat cut-and-fill.

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11 - After Bicknell, H. L., Rill Stoping in the Butte District: Compressed Air Mag., vol. 26, July, 1921, pp. 10153-10158.

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Labor efficiencies in stopes at Butte[1]

Stoping method	Tons per man-hour[2]
Square-set	0.6313
Rill (untimbered)	..8017
Rill (timbered)	.7796
Back filling	.6106
Stull	.5075
Average, all stopes	.6515

1 - Daly, W. B., and others, Mining Methods in the Butte District: Trans. Am. Inst. Min. Eng., vol. 72, 1925, p. 282.
2 - Includes all men in stopes and includes sorting.

If a considerable amount of wall prospecting must be done to discover possible parallel veins or ore shoots, the horizontal stope provides better opportunities to do this work. Where this prospecting is done by drifting or raising, the disposal of the waste would complicate mining in a rill stope. If it is done with long-hole compressed-air drills or with diamond drills, the working conditions along the slope of a rill would be found unfavorable.

Local custom is undoubtedly often an important factor in making the choice between the two methods of stoping. Miners as a rule favor the old, tried method of doing their work. Square-setting, shrinkage, and open stopes fall under this description. From shrinkage stoping to horizontal cut-and-fill is not a great change, as far as working conditions are concerned, and even obviously increases the safety of the men. Rill stoping, however, introduces a different set of conditions in that much work is done on a sloping floor and the men must be continually conscious of what is being done above them in the stope. For these reasons local prejudice usually favors the former method.

Size of Stopes or Stope Sections

In small orebodies a single horizontal cut-and-fill stope is frequently extended over the entire ore area. This has often been the case at Questa, Copper Queen, Pilares, and other mines. Such a procedure has a distinct advantage in that no gob fences have to be built and the inconvenience of mining alongside old fills and the cost of pillar extraction are likewise avoided. Several crews can work in a long stope in a vein without sectionalizing the stope by starting work at two or more places in a roughly horizontal back or by advancing the mining faces from end to end in a series of stopes. The latter process is indicated in Figure 1,K, which illustrates one type of Michigan copper range practice.

A small orebody is likewise often extracted completely by an inclined cut-and-fill stope operated as a single rill, or perhaps a double rill worked by one mining crew alternating from side to side.

A long orebody in a vein could be worked conceivably by advancing a single rill after it was developed from one end to the other, thus eliminating the cost of driving fill raises except the one needed to start the work. One serious objection to this would be the relatively low rate of production from the orebody and the complete stopping of production while filling the stope after each cut. Another objection would be the extreme length of the rill. The length on a 40° slope would be 140 feet from the top of an extraction drift to the floor of a drift 90 feet above.

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In long veins, therefore, if it is necessary to have more than one rill face working, stoping sections must be laid out for separate operation. A section may consist of a fill raise in the middle and two outward rills extending to chutes at either end, or, more commonly, may comprise two rills sloping down from fill raises at either end to a common extraction chute at the center of the section. Such double sections are usually 125 to 150 feet long. Fences or bulkheads are built at the ends of sections to retain the fill in order that adjoining sections may follow at a lower elevation. The waste raise may be of the stulled type used at the Eighty-Five Mine,[12] in which case, in a narrow vein, it will in itself serve to retain the waste fill in the higher of the two stope sections.

In a wide orebody, such as was worked by longitudinal rill stopes on the 700 level of La Colorada mine, Cananea (fig. 1, B), it is often necessary to build heavy cribbed bulkheads or gob fences across the stope in order to run fill into one or the other of the two rills which are served by the same fill raise. It is also necessary to prevent the fill from overrunning the extraction chute at the bottom of the slope, where the ore from the opposite rill is being drawn out; further timbering expense is incurred in this way.

When an orebody is of such large size or the ground so heavy that it is apparent the stoping will have to be carried on in sections less than the full size of the orebody, there are two possible procedures. Either the cut-and-fill method will be used exclusively or an initial line of stopes will be mined by this method and pillars left between for later extraction by some other method, such as square-setting or the Mitchell slice.

In the first case, stope sections 15 to 25 feet wide are usually (as at the Pecos mine, fig. 1,I) laid out across the strike of the deposit, that is, from wall to wall. Such sections may be carried in a stepdown series, each one being a sufficient distance below the older adjacent stopes to prevent interference of operations. Moreover, the face of one stope is carried far enough above the next in line to allow time for the fill to settle and afford maximum support to the walls while the second is being mined. For the sake of efficiency of operations, the width of each section should be as much as the ground will permit.

In the second case, that is, if a stope and pillar system is adopted, the widths of stopes and pillars depend on several factors, which can be appraised only roughly in advance of actual experience. Some of these factors are the ability of the ore to stand over wide spans, the amount of wall pressure developed by mining, and the relative cost of the methods proposed for mining the original stopes and the pillars. The natural tendency is to make the stopes as wide as the ground will stand safely, and the pillars as narrow as will support the walls without complete failure. In this way it is expected to get the greatest advantage of the lower cost method. Unfortunately, the pillars are often made so narrow that by the time they are mined the ore in them is taking weight and may even be very badly fractured. In this event the mining cost in the pillars will be increased seriously, resulting in a high average cost for the combined stopes and pillars. At La Colorada mine horizontal cut-and-fill stopes were laid out 30 feet wide and the intervening pillars 40 feet wide on one level and 50 feet on a lower level. The pillars are to be extracted by top slicing. In large orebodies at Pilares rill stopes not over 30 feet wide are laid out across the strike of the orebody. The width of pillars is not stated; they are extracted by square-setting or top slicing. At United Verde, in the main orebody, vertical pillars about 50 feet wide have been left at intervals of 150 to 200 feet, extending across the orebody and continuous from the 1500 level down. Even though conforming to this pillar system, the stopes are very irregular in area and range from 30 to 160 feet across the ore body and from 60 to 200 feet along the strike. This variation is due both to the irregularity of the orebody itself and

12 - Youtz, Ralph B., Mining Methods at the Eighty-Five Mine, Calumet and Arizona Mining Co., Valedon, N. Mex.: Inf. Circ. 6413, Bureau of Mines, 1931, pp. 7-8 and Fig. 6.

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to the varying ground conditions, which sometimes prohibit wide backs. The cut-and-fill stopes, however, if the ore is continuous, are not mined in sections, that is, with lengths less than the distance between pillar lines. In exceptional cases one end of a small offset section may be left to be extracted later by some timbered stoping method. The main pillars, both vertical and horizontal, are mined either by square-setting or by top-slicing with back filling to prevent subsidence. Thus cut-and-fill stoping is in no case carried on adjacent to filled stopes. At the Magma mine stopes are 16 feet wide and separated along the strike of the vein by 14-foot pillars, which are later mined by the Mitchell slice method.

No mine is known in this country where the cut-and-fill method has been used for extracting pillars, although it is used for this purpose in some of the mines in the Krivoy-Rog district in Russia.[13] Lavender,[14] however, states: "In laying out the stope and pillar sections, it was planned to work both stope sections by cut-and-fill methods. The pillar was to be recovered by means of a Mitchell slice running at right angles to the long stope dimensions. While work at present has not progressed to the point of mining the pillar sections, the success of the cut-and-fill method in this section (of the mine) has made it probable that a serious attempt will be made to use the cut-and-fill in mining the pillars." At this mine the stopes were laid out in pairs, each stope 45 to 50 feet wide, and each pair of stopes separated from the next pair by a pillar 45 feet wide. In some places in this orebody where the distance from hanging wall to footwall was excessive and would have resulted in a very long rill, each stope was subdivided further by a gob fence or single lead of square sets into two equal parts.

Silling Out

The relative merits of silling out a stope on the level or above it are the subject of much discussion. Two factors favoring silling out above the level are the decreased cost of handling the ore, which can be loaded through chutes into cars instead of shoveled by hand, and the lowered cost of maintaining the haulage drift, which is protected by pillars of solid ore above or at least on both sides of it. When stopes are thus silled out above the level, the extraction drift may not require timbering, as, for example, at Cananea.

Factors which favor silling out at the elevation of the floor of the drifts are: The possible better location of the gangway and chutes with respect to the orebody as determined by the silling out; the avoidance of the often difficult and costly work of extracting pillars sometimes badly crushed, beside and over the drift upon the completion of the next stope below; the elimination of the chute raises from the drift to the sill of the stope; and the possibility of abandoning fill raises upon the completion of the stope they were driven to serve, instead of having to maintain them to provide filling for these pillars. In some cases the last factor may be of little importance because fill can be obtained at this stage of the operation by tapping the waste fill in the stope above.

At Magma the disadvantage of shoveling the sill floor ore by hand is avoided by using mechanical shovels; this permits a very rapid opening of the sills. At Matahambre it is found that it is very difficult to maintain the haulage drift while removing the pillars under large stopes, so that if there is not already a drift in the footwall, one is usually driven at that time.

Sill floors are opened either by advancing a face the full width of the vein from one

13 - Ayvazoglou, W., The Method of Underground Mining of Iron Ore in the District of Krivóy Rog. (Translated from the original Russian of A. K. Bouldovsky): Inf. Circ. 6254, Bureau of Mines, March, 1930, 48 pp.

14 - Lavender, H. M , Mining Methods at the Campbell Mine of the Calumet & Arizona Mining Co., Warren, Ariz., Inclined Cut-and-Fill and Semi-Shrinkage: Inf. Circ. 6289, Bureau of Mines, April, 1930, 18 pp.

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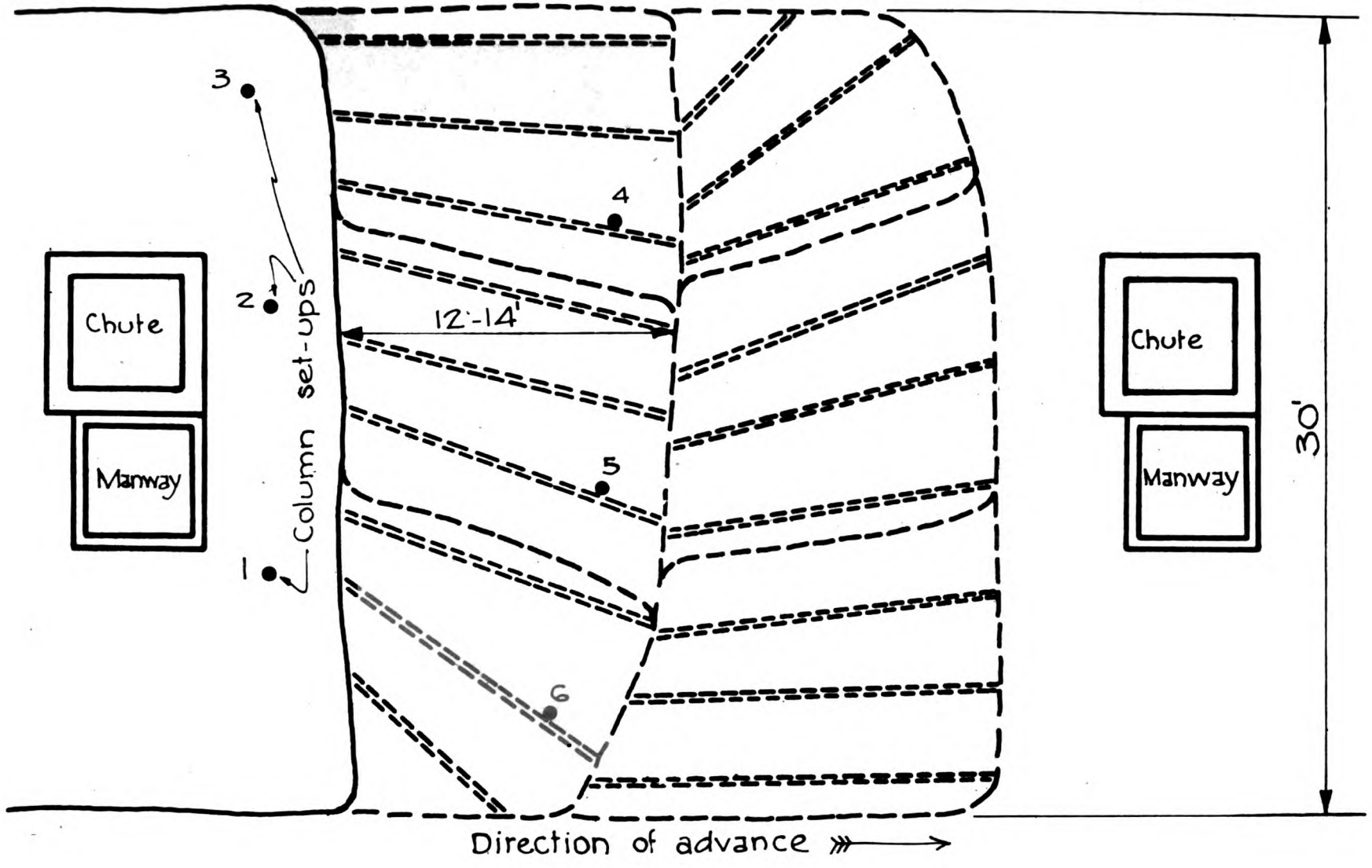

Figure 4.— Drill round for silling out cut-and-fill stopes, La Colorada mine, Cananea

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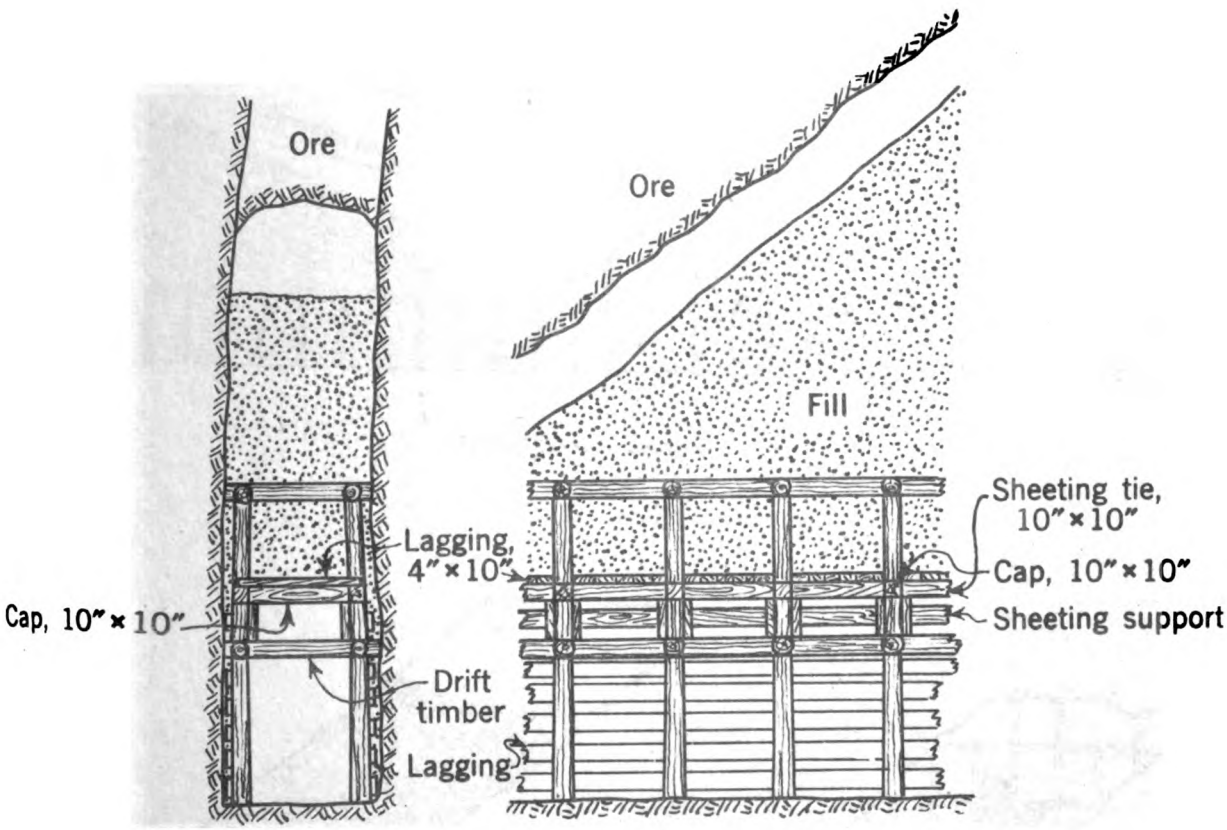

Figure 5.— Gangway timbering af Butte, showing use of sheeting

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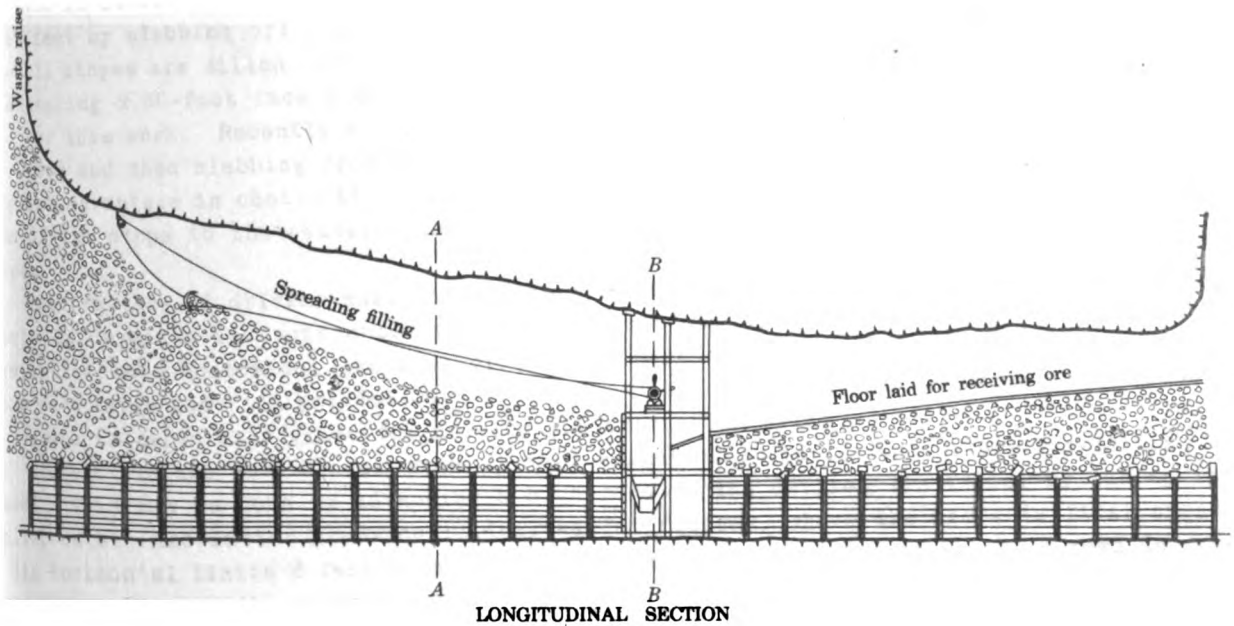
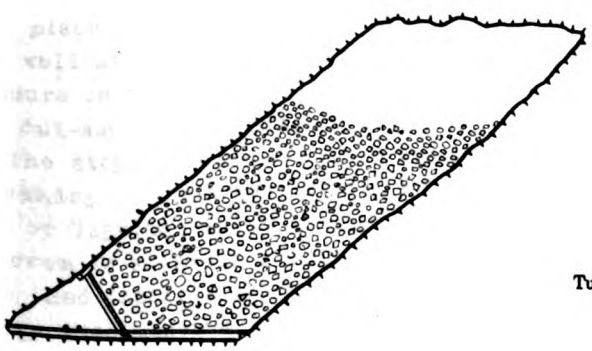
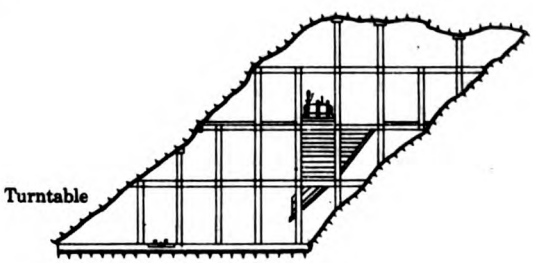


LONGITUDINAL SECTION



SECTION *A-A*



SECTION *B-B*

Figure 6.– Cut-and-fill stope at Park Utah mine, showing novel sill or gangway timbering

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end of the panel to be mined to the other, or by slabbing off the walls of a drift. The latter is the usual practice at mines where the stope is silled on the level. At Magma, in addition to a drift in ore along the footwall, crosscuts are run from this drift to the hanging wall at 30-foot intervals; the sill of each stope is then opened up to its full length of 16 feet by slabbing off the sides of this crosscut. In La Colorada mine at Cananea cut-and-fill stopes are silled either 11 or 22 feet above the level (one or two top-slice floors) by advancing a 30-foot face from end to end of the stope. Figure 4 shows a slabbing round used for this work. Recently a stope was opened by driving a drift along the center line of the stope and then slabbing from the drift to full sill dimensions. This was found to offer no clear advantage in cost. At the Eighty-Five mine small sublevel drifts are run from both ends of the stope to the center raise and then widened by slabbing to form the sill of the stope.

The practice of driving sublevel drifts preparatory to silling out has several advantages: It develops the vein quickly, gives several points of attack for silling out, and permits the most economical handling of a large part of the sill ore, especially when the stope is a large one. As pointed out previously, this drift may even be extended far enough in advance to serve as a guide for driving the extraction drift.

Stope sills are usually cut to an initial height of 10 to 15 feet before any filling is placed. This may be done by advancing a high face to the desired limits, or by taking two smaller cuts. The latter practice is followed at Pilares, where the stope is first opened to its horizontal limits 8 feet high, after which an additional 6 feet is taken down and removed before placing the gangway and filling.

Building Gangways and Chutes

If a stope is silled on the level, timbering is necessary to provide a gangway after the filling is placed. In a narrow vein this timbering may consist merely of caps set in hitches in the wall at the desired height above the floor. If the walls are weak or the anticipated pressure on the caps great, one or two posts may be stood under each cap.

At Butte, cut-and-fill stoping is confined to the narrower veins, not over 15 feet wide. In such veins the stope is always silled out on the level to the full width of the ore. To prevent the breaking of drift caps by the weight of fill it is the practice to carry the fill on "sheeting," or lagging, which is laid on false sets about 30 inches above the drift caps (*fig.* 5)[15] or even to maintain a full set open above the drift. This greatly reduces drift timber repairs, and facilitates them if unavoidable.

At the Park Utah mines a novel gangway timbering method is used when ground conditions are favorable (fig. 6). With a good back this method of timbering should result in an appreciable saving in timber cost. It is comparable to the use of stulls from wall to wall in narrow vein stopes in place of regular drift sets.

In wider veins several methods of sill timbering are used. One method is to stand a line of regular drift sets along the desired gangway position, lag the back and sides securely, block the sets upright by temporary props or stulls, and fill the rest of the sill with waste. Square-sets are used frequently in place of drift timber. At some mines the gangway sets are stood on sills of 5 by 10 timber. This is advantageous if the gangway must be maintained while mining out the ore below it.

If the size and life of a stope will justify the slight added cost, pony sets should be provided at each chute. This practice will invariably increase the speed and safety with which cars can be loaded. If the ore is to be hand trammed in small cars, the economy of

15 - After Bicknell, H. L., Rill Stoping in the Butte District: Compressed Air Mag., vol. 26, July, 1921, pp. 10153-10158.

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loading from a pony set is doubtful. In this event the drift should be wide enough at the chute to give the loader safe working room, and preferably high enough so that a platform can be built on the side of the drift for the loader to stand upon.

Methods of Breaking Ground

Ore is broken in a horizontal cut-and-fill stope by either breast stoping or overhand stoping. In the former the drill holes are practically horizontal; in the latter the holes are drilled vertically, using the stoper type of drill.

The choice between the two methods should depend partly on the nature of the ground. Granting that the aim of the primary blasting is to reduce as much as possible of the ore to a size to be handled conveniently out of the mine with a minimum of secondary breaking, advantage should be taken of the joint structure of the ore when pointing the drill holes. If such jointing is horizontal, causing a tendency for large masses to fall with the first hole of a round of shots, the use of vertical holes will counteract this tendency. Against this argument several factors may be pointed out which strongly favor the use of breast stoping with horizontal holes, regardless of the ground structure. It is often dangerous for the miners to stand under the back being drilled, and especially under a brow such as exists at an advancing stope face. When breast stoping, the miner stands in front of the dangerous brow and under a fresh, recently trimmed and presumably properly arched back. Vertical holes undoubtedly scatter the ore more than the horizontal holes, making more work for the shovelers. Finally, stoper holes pointed into the back usually leave a more uneven and ragged back after blasting, which is harder to trim down to a safe condition and which is more difficult to support, if support is required. Formerly, breast stoping was favored in very hard ground because the Leyner-type machines could drill such ground faster than the stoper drills. This factor is of less importance to-day because of the development of larger and more powerful types of stoper machines. Incidentally, the use of stoper machines prohibits the filling of the stope closer than 5 or 6 feet from the back, which in certain cases might be considered a disadvantage. At the United Verde it was found by trial that whereas the use of stopers gave better fragmentation than was possible with horizontal holes, the practice had so many undesirable features that it had to be abandoned. That the advantage lies with breast stoping in most cases is confirmed by the fact that in a large majority of the mines using cut-and-fill methods (see Table 3) the ground is broken in this way. Only the United Eastern, Park Utah, Santa Gertrudis, and to some extent the Pecos and Matahambre mines of the group considered in this article use back stoping in horizontal stopes. This is rather striking in view of the almost universal acceptance of the method in shrinkage stoping.

The height of a single cut ranges from 6 feet, as in the United Eastern and Matahambre mines, to 11 feet, as at Cananea. Filling is done usually after each cut, but at some mines two cuts may be taken before filling, the second being drilled from on top of the ore broken by the first. Economies in breaking ground doubtless can be achieved in many mines by the use of high lifts; such measures are especially advantageous when scrapers are used to move the broken ore and spread the waste fill, because scrapers are most efficient when large quantities of material are available. This is exemplified by the practice at Park City and recently at Cananea. At the latter mine the practice of breaking two 11-foot slices has resulted in a great decrease in stoping cost. However, an important accident hazard is certainly increased when higher backs are permitted over places where men have to work. A back not over 10 or 12 feet high is easily inspected and barred down. If higher than that, the back is usually "out of sight, out of mind."

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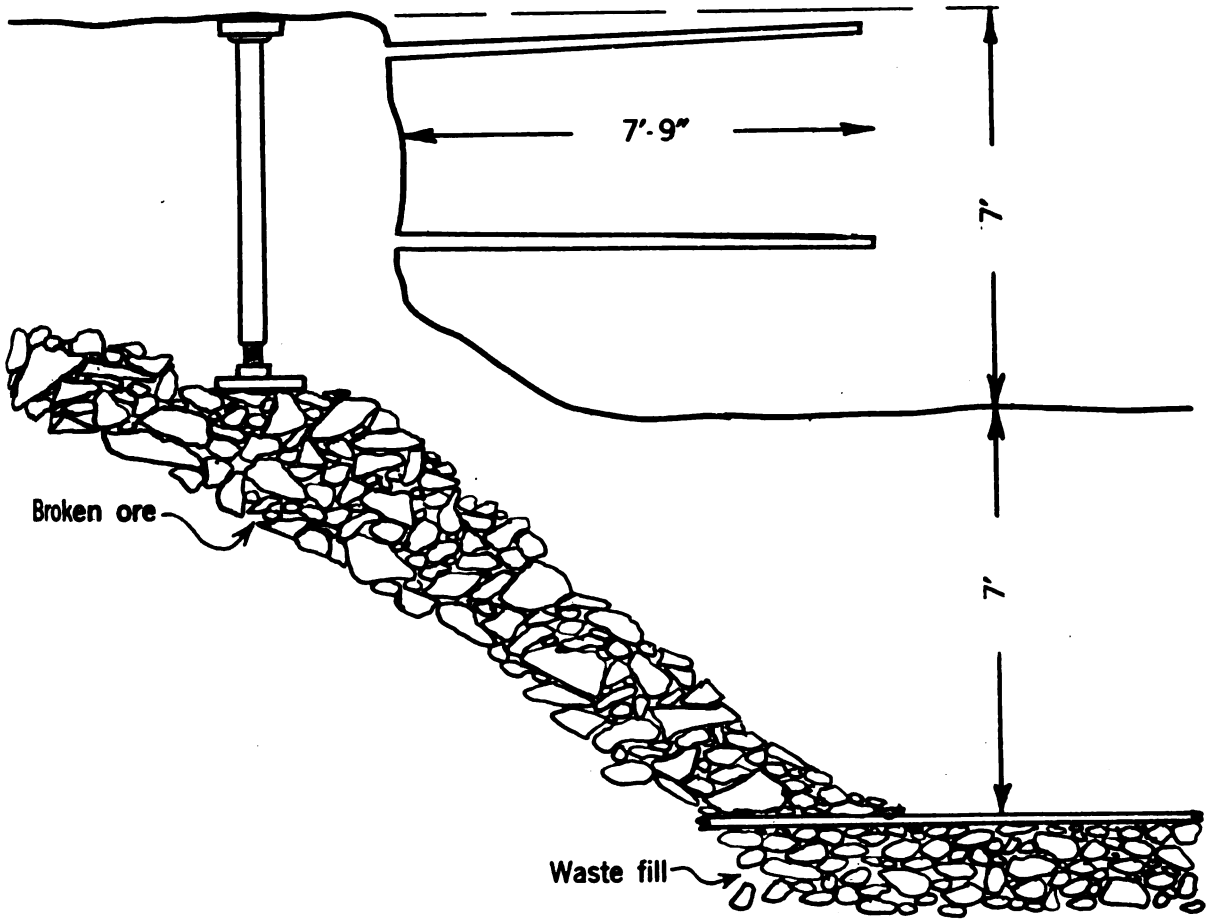
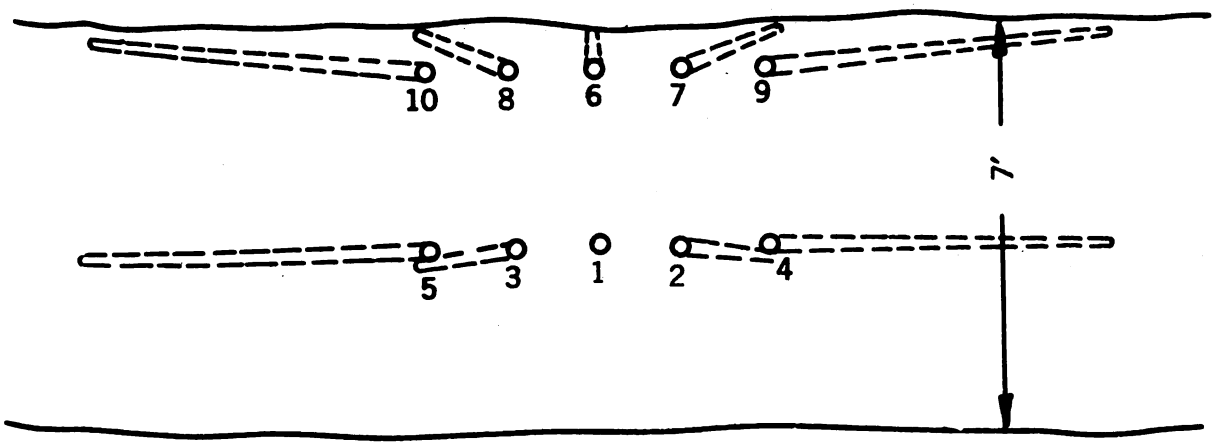


SIDE ELEVATION



FRONT ELEVATION

Figure 7.— Drill round in cut-and-fill stope, United Verde mine

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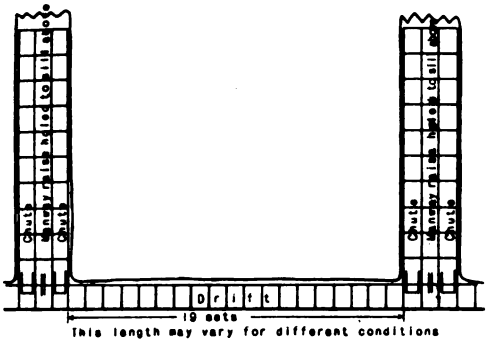
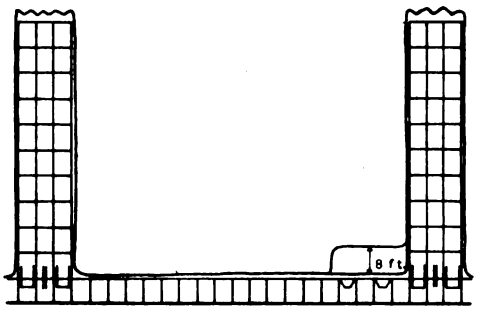
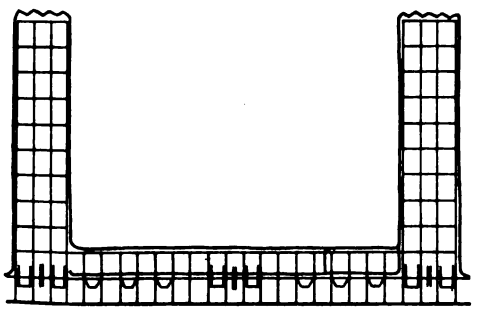


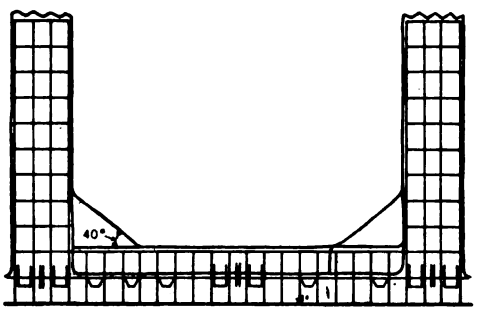
A, Raises and drifts timbered for double rill



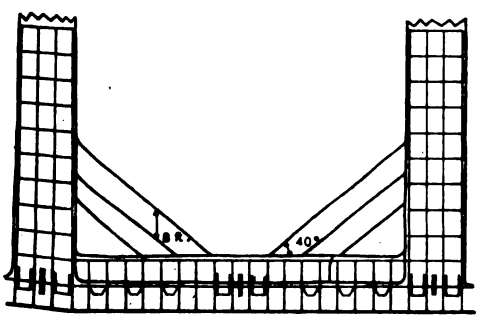
B, First cut above sill. Temporary slides are used for this and the first rill cuts



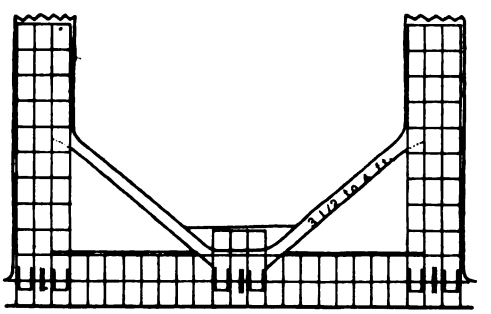
C, First floor or sheeting floor timbered. Stope chutes and manway made ready



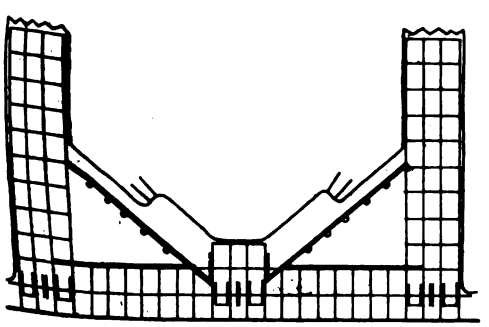
D, Cuts near raises



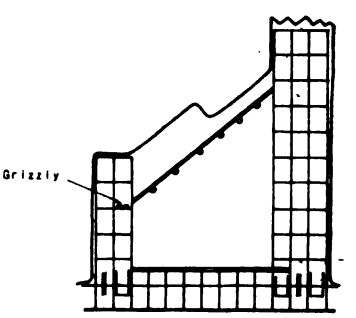
E, Cuts to stope chutes. Filling and floor may be necessary prior to this condition



F, Filling and slides with stope chute sets and manway timbered before starting the first full cuts



G, First full cuts. Filling, slide, and ore on slide controlled by stop boards. Drilling is done parallel to slide from ore pile



H, Single rill stope, sheeting, filling, slides, stop boards, and ore pile

Figure 8.- Progressive stages of rill stoping at Butte

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Drill rounds for cut-and-fill stopes seldom are standardized, because of the irregularities of face and ground conditions usually existing in even a single stope. A common practice in breast stoping is to drill two horizontal rows of holes, each row having a burden of 3 to 5 feet. The holes are fanned out from the position of the drill set-up so that each hole will have about the same average horizontal burden of 3 to 5 feet. Figure 7 is a typical stope round used at the United Verde mine.[16] Figure 8, which illustrates several stages in the development of a rill stope at Butte, shows that the ground is broken by breast stoping, although earlier practice was sometimes to drill the whole length of the rill with stopers before starting to blast.

Drill rounds for back stoping, using stoper drills, are similar to those used in shrinkage stoping. In wide stopes the holes should be spaced regularly in parallel straight rows in order to break the ground to a smooth face.

Drilling

Wide stopes, hard ore, and the use of the horizontal cut-and-fill method of mining obviously favor the use of mounted drifter-type drills. Narrow veins, softer ore, and the difficulty of setting up drifters on the slopes of a rill are factors favoring the use of stoper drills or even jackhammers. A great many other factors are likewise of importance, such as the initial and repair cost of different types and sizes of drills, drilling speeds, depth of hole desired, the aptitude of the miners, and the cost of labor.

Stopers can be used in a cut-and-fill stope only when the ore is of such a nature as to form a strong secure back over the required span. Otherwise the danger to the miner operating the drill is excessive. At Matahambre, stopers are used only to make the initial cut in a back; the cut is then carried along the stope by drilling flat holes in the breast with jackhammers. Cananea, United Verde, and Calumet & Arizona use heavy drifter-type machines in stopes. At Pilares and in the Copper Range mines mounted jackhammers are used.

The use of wet drills is universal in many districts, less common in others, and infrequent in a few. Where stopers are used it is particularly difficult to force miners to drill "wet," because of their natural objection to the resulting dirt and water, especially in cold mines. Dry drilling raises dust which is almost always harmful to breathe, and the use of water is believed to remedy the condition in large part without seriously increasing drilling costs.

Drill Steel

Hollow round steel 1 or 1¼ inches in diameter is generally used with machines of the drifter type. One-inch, hollow quarter-octagon steel is the most common for stopers and 7/8-inch hexagon for jackhammers. Bits are usually of the cross form, with double tapers of 5° and 14° on the sides of the wings. The purpose of the double taper is to present as much material as possible to bear on the side of the hole, thus reducing loss of gage. Length changes range from 10 to 22 inches, and the gage is generally reduced 1/16 or 1/8 inch with each change in length.

16 - From Mills, C. E., Drilling and Blasting Underground at United Verde Mine: Eng. Min. Jour., vol. 121, May 29, 1926, pp. 877-881.

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Table 3.- Comparative data on mining details at mines using the cut-and-fill stoping method

(Section A)

General data						
Name of mine	Principal metals	Approximate grade	Ore mined daily, tons	Types of orebodies	Cubic feet per ton ore in place	Percentages o different sto ing methods us
	1	2	3	4	5	6
Campbell	Copper	5 per cent	1,500	Solid, large sulphide mass, well defined footwall dipping 25° to 90°; 50 to 250 by 500 feet in plan	9	FR, semi-SHR, S MS.
Copper Queen	Copper	5 per cent	1,000 excluding porphyry ores	Irregular, small to large limestone replacement, sulphide and oxide	9-11	SS 55, TS 40, H and FR 4.
United Verde	Copper	5 per cent	3,000	Irregular, small to large replacements in schist; main orebody pipe-like, about 300 by 1,100 feet, 60° dip	8-10	HCF 61, FR 3, S 27, SHR 1.
Frood	Copper, nickel	4 1/2 per cent copper 2 1/2 per cent nickel	6,000?	Large tabular massive sulphide body in greywacke, 40 to 200 feet wide, 65° dip	Probably 9	HCF, minor amou SS.
La Colorada	Copper	(?)	1,500	Massive sulphide pipe, about 200 by 300 feet	8-12	FR 40, HCF 14, 12, SHR 14, d velopment wor
Pilares	Copper	3 per cent	3,000	Irregular, small to large, usually tabular and vertical sulphide bodies in breccia, around and within a large pipe like structure	12	HCF 60, FR 18, 18, TS 3, SHR
Magma	Copper	7 per cent	800	Large, continuous vein, 80° dip, 5 to 60 feet thick, averaging 25 feet, 1,200 to 1,500 feet long	10 1/2	FR 35, MS 40.
United Eastern	Gold	$20	300	Quartz-calcite veins in fault fissures in andesite, 500 to 1,500 feet long, 500 to 1,000 feet deep, up to 45 feet thick, dip 70°-75°	(?)	All HCF.
Champion	Copper	2 per cent	1,500	Tabular deposits of native copper in amygdaloid bed dipping 70°, maximum thickness 80 feet, average 17 feet	11	Formerly chief HCF, now chan to FR on lowe levels.
Butte District	Copper, silver	4 1/2 do.	10,000	Quartz-sulphide veins in granite, generally steep-dipping, averaging 10 to 30 feet thick	9-11	SS 80, HCF and 15.

1 - Symbols are used to represent stoping methods as follows: HCF - Horizontal cut-and-fill. FR - Inclined cut-and-fil filled rill. OS - Open stope. STS - Stulled stope. SHR - Shrinkage. SS - Square-set. TS - Topslice. M - Mitchell s

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Table 3.- Comparative data on mining details at mines using the cut-and-fill stoping method - Continued

(Section A - Continued)

General data						
...e of mine	Principal metals	Approximate grade	Ore mined daily, tons	Types of orebodies	Cubic feet per ton ore in place	Percentages of dif ferent stoping methods used[1]
	1	2	3	4	5	6
...os	Zinc, lead, copper	16 per cent zinc 4 do. lead 1 do. copper	500-600	Vein-like, lenticular sulphide replacements in shear zones in diorite schist, average 25 feet thick dipping 70° to 90°	(?)	SS 50, HCF 40.
...k Utah	Lead, zinc, silver	Relatively high grade	600	Lenticular ore shoots in fissures in limestone, 3 to 80 feet thick, 40° to 55° dip	(?)	Chiefly SS. HCF and some STS used only in stopes 12 feet or less in width.
...ahambre	Copper	5 per cent	1,000	Irregular tabular or pipe-like sulphide orebody. Quartzite footwall, shale hanging wall, dip generally 42° to 45°, average thickness perhaps 30 feet	10-11	HCF.
...n Cap	Copper	3 do.	300	Sulphide and oxide ores in fissure vein, dipping 80°, thickness 3 to 40 feet, average mining width probably 5 feet	11	FR.
...lluride ...istrict	Gold, silver	$5 to $8	150-400	Quartz-sulphide fissure veins in volcanic rocks, dip 60° to 80°, average thickness 2 to 3 feet	11 3/4-13	Various, including cut-and-fill.
...hty-Five	Copper	2 1/2 per cent	300	Quartz-sulphide veins in fault fissure in granodiorite and andesite, average width 5 feet, dip 80°	Probably 12	SHR 85, FR 15 (approx.).
...ound Hog	Zinc, lead, copper, silver	15 per cent zinc 8 do. lead 3 1/2 do. copper 6 1/2 ounces silver	130	Quartz-sulphide veins in fault fissure in granodiorite and diorite porphyry, dip 50°, thickness 3 to 25 feet	9	SS, FR, a little HCF.
...esta	Molybdenum	7 per cent	30-50	Narrow, irregular, quartz-sulphide veins in alaskite porphyry. Ore averages 12 to 18 inches in width. Dips range from 20° to 90°, averaging 60°	(2)	HCF 90, OS 10.
...ata ...ertrudis	Silver, gold	13 ounces silver, $1.50 gold	1,000	Large and persistent ore shoots in silicified fissure zones in andesite. Economic mineral argentite. Widths up to 18 feet. Main vein dips 60°	13.6	HCF, SS, SHR.

[1] Symbols are used to represent stoping methods as follows: HCF - Horizontal cut-and-fill. FR - Inclined cut-and-fill, or filled rill. OS - Open stope. STS - Stulled stope. SHR - Shrinkage. SS - Square set. TS - Top slice. MS - Mitchell slice.

[2] The screened and sorted ore sent to the mill averages 2.56 specific gravity, which is equivalent to 12.5 cubic feet per ton in place, or 19 cubic feet per ton broken.

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Table 3.- Comparative data on mining details at mines using the cut-and-fill stoping method - Continued

(Section A - Continued)

Name of mine	General data: Principal metals	Approximate grade	Ore mined daily, tons	Types of orebodies	Cubic feet per ton of ore in place	Percentges o different sto in methods use
	1	2	3	4	5	6
Lucky Tiger	Silver, gold	40 ounces silver, $5 gold	250	Sulphide veins in fracture planes in rhyolite or rhyolite tuff. Average thickness 1.7 feet, rarely as much as 20 feet	11.5	Chiefly HCF, s FR, minor amo of SHR and OS
Zaruma	Gold	$9	300-350	Quartz-calcite-sulphide gold-bearing veins in fault fissures in altered dacite; thickness 3 to 20 feet, average dip 65° to 70°	12	FR.

(Section B)

Name of mine	Development details: Depth mined, feet	Level interval, feet	Location of extraction drifts and crosscuts	Size of drifts, feet	Proportion of drifts timbered	Silling out stopes: Stopes silled on or above levels	Height of s cut, fe
	7	8	9	10	11	12	13
Campbell	2,300	[2]100	Drifts in hanging wall close to ore. Crosscuts at pillar lines	6 by 8 motor 5 by 7 other	Only under stopes	10 feet above	-
Copper Queen	1,800	100	Usually in ore, but various due to irregular shape of orebodies	5 by 7	Small, previous to silling out stope	10 feet above	10
United Verde	3,000	150	In country rock close to ore	[3]5 by 10	About 5 per cent timbered when driven	21 to 25 feet above	14 (two 7- cuts).
Frood	3,000	200	Footwall drift 65 feet from ore. Extraction drifts parallel to strike on 44 or 65 foot centers	(?)	(?)	30 feet above	7
La Colorada	1,500	125	Around and through orebody at regular intervals (generally 40 feet)	5 by 8	About half	22 feet above main levels; 11 feet elsewhere	11
Pilares	1,900	100 and 133	In ore	5 by 7 and 6 by 8	Small	On or 20 feet above	14 (one 8 one 6).

1 - Symbols are used to represent stoping methods as follows: HCF - Horizontal cut-and-fill. FR - Inclined cut-and-fil filled rill. OS - Open stope. STS - Stulled stope. SHR - Shrinkage. SS - Square-set. TS - Top slice. MS - Mit slice.

2 - Alternate levels are used as haulage levels.

3 - Gangways under stopes. Elsewhere timbered drifts are 5 by 7 and untimbered drifts 6 by 8.

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Table 3. Comparative data on mining details at mines using the cut-and-fill stoping method - Continued

(Section B - Continued)

ne of mine	Development details					Silling out stopes	
	Depth mined, feet	Level interval, feet	Location of extraction drifts and crosscuts	Size of drifts, feet	Proportion of drifts timbered	Stopes silled on or above levels	Height of sill cut, feet
	7	8	9	10	11	12	13
na	2,800	[1]100-150	In footwall 25 feet from vein, and drifts in ore at footwall and crosscuts through vein	7 by 9 and 8 by 8	Large ?	On level	10-12
ed Eastern	1,300	75, 100, 150, 200	In vein	5 by 7 1/2	Little	On level	Either one or two sets plus 6 feet.
pion	4,000	100	In lode	8 high, 9 to 13 feet wide	Nearly all under old system. Only a small part at present	On levels and sublevels	8
e district	3,600	100, a few 200	In veins	5 by 7 1/2	Small, in advance of stoping	On level for veins under 15 feet	(?)
s	1,050	100	In ore at footwall	5 1/2 by 7 drifts 7 by 8 crosscuts	All drifts in ore timbered	20 feet above	7 ?
Utah	600	150	In ore near either wall, or in heavy ground a drift in country rock on both sides of orebody	8 by 8 main drifts; 5 by 7 crosscuts	(?)	On level	9
ambre	1,800	100-130 To be 150 in future	Main drifts in footwall Extraction drifts in ore	5 by 7	Small	14 feet above	12
Cap	1,540	100	In ore on footwall	5 by 7	None, until stopes started above	On level	15
ride trict	500-1,000	100-250	In ore	6 by 7, 6 by 9	(?)	do.	(?)
y-Five	1,600	100	In ore, on footwall if vein is more than drift width	5 by 7 or 6 by 7	(?)	20 feet above	Cut to inclined back.
d Hog	500	[1]50	In ore	5 by 7	Little, until stopes are silled out	On level	14
a	(2)	40-100	In ore	4 1/2 by 6 1/2	Abcut 10 per cent	do.	7-12
rudis	2,000	165	In footwall	(?)	(?)	do.	One floor.
Tiger	1,200	100	In ore	5 by 7	(?)	do.	(?)
a	1,100	98	In ore, and in footwall where necessary to avoid heavy ground or water	5 by 7	Very little until stopes are silled out	do.	16 1/2

lternate levels are used as haulage levels.

ll development is by adits. The total vertical range of the principal workings is about 500 feet.

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Table 3.- Comparative data on mining details at mines using the cut-and-fill stoping method - Continued

(Section C)

Name of mine	Breaking ground						Moving ore out of stope			
	Horizontal or inclined stoping	Breast or back stoping	Height of cut, feet[1]	Depth of holes, feet	Kind of drills	Percentage of ore requiring secondary breaking in stopes	Percentage of broken rock sorted out as waste	Method of moving ore in stope	Center to center spacing of chutes, feet	Grizzl opening inche
	14	15	16	17	18	19	20	21	22	23
Campbell	Inclined	Breast	12	7-8	Drifters	Considerable ?	None	Gravity[2]	10	10
Copper Queen	Inclined	do.	10	5-7	do.	About 10	None	do.	Various	10
United	Horizontal	do.	7	7-8	do.	Large	2	Hand shoveling	16 1/2 or 22	11
Verde	Inclined	do.	10	(?)	(?)	do.	2	Gravity		
Frood	Horizontal	do.	7	(?)	Drifters	(?)	Probably none underground	Mostly hand shoveling, some scrapers	16 1/2 or 27 1/2	(?)
La Colorada	Inclined	do.	11	10	do.	About 30	None?	Gravity and scrapers in rills. Hand shovels, wheelbarrows, and scrapers elsewhere	150 to 200 in rills, 40 in HCF	10
	Horizontal	do.	11	12						
Pilares	Both	do.	8	(?)	do.	(?)	Always some	Hand shoveling and gravity	30	12
Magma	Inclined	do.	7	(?)	Stopers	Little	Probably none	Gravity	30	9
United Eastern	Horizontal	Back	6	(?)	do.	Little, probably	Probably none	Hand shoveling	22 1/2	-
Champion	Both	Breast	4	10	Drifters	(?)	40	Gravity and hand shoveled into cars	Approx. 200	-
Butte district	Both	Back, in both horizontal and rill	8	6	Stopers	(?)	Approx. 20	Hand shoveling or gravity	25 to 40 in HCF, 112 in rills	(? -
Pecos	Horizontal	Both?	7	(?)	Stopers, jackhammers	Probably very small?	20	Hand shoveling	15 to 30	10
Park Utah	Horizontal	Back and breast?	16 (two 8-foot cuts)	8	Drifters, stopers	Very small	None, probably	Scrapers	50 to 150, average 100	(?

1 - Thickness of cut normal to slope in rill stopes.

2 - A small amount of hand mucking is done. Scrapers are used in silling out stopes.

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Table 3. Comparative data on mining details at mines using the cut-and-fill stoping method - Continued

(Section C Continued)

Name of mine	Breaking ground						Moving ore out of stope			
	Horizontal or inclined stoping	Breast or back stoping	Height of cut, feet[1]	Depth of holes, feet	Kind of drills	Percentage of ore requiring secondary breaking in stopes	Percentage of broken rock sorted out as waste	Method of moving ore in stope	Center to center spacing of chutes, feet	Grizzly openings, inches
	14	15	16	17	18	19	20	21	22	23
lahambre	Horizontal	Chiefly breast?	6-7	(?)	Jackhammers some stopers	(?)	Some sorted out, paid for by the ton	Hand shoveling or wheelbarrows	50	(?)
n Cap	Inclined	Breast	6 ?	(?)	Stopers	(?)	None	Gravity	125	(?)
luride	Horizontal	(?)	(?)	(?)	(?)	(?)	(?)	(?)	50	-
strict										
hty-Five	Inclined	Breast or back	5	(?)	Stopers	Almost none	None	Gravity	140	10
und Hog	do.	Breast	3	(?)	Stopers, jackhammers	Almost none	Considerable	Gravity and hand shoveling	50	-
sta	Horizontal	Back	(2)	Seldom more than 3 feet	Stopers or hand drills	Almost none	About 60	Hand shoveling, rarely wheelbarrows	50	-
ta	do.	Back	Probably 6	(?)	Probably stopers	(?)	None, probably	Hand shoveling and wheel barrows	165	6
ky Tiger	Both	Chiefly back	(?)	(?)	Chiefly stopers, some hand drilling	Small	Considerable	Hand shoveling and wheelbarrows; gravity in rill stopes	50 in flat backed stopes, 100 in rall stopes	(?)
ma	Inclined	Breast	4 (approx.)	6±	Stopers	About 10	Little	Gravity; some hand shoveling	(?)	(?)

[1] Thickness of cut normal to slope in rill stopes.

Drilling and breaking practice in stopes varies greatly because of the irregularity of the veins.

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Table 3.- Comparative data on mining details at mines using the cut-and-fill stoping method - Continued

(Section D)

Name of mine	Flooring: Sill flooring	Flooring: Regular flooring (temporary)	Filling: Source of filling material	Filling: Method of spreading fill in stope	Filling: Filled how close to back, feet	Stope crews	Methods of payment
	24	25	26	27	28	29	30
Campbell	Scrap timber, sometimes laid on sills	2 by 10 or 12 inch, 10 feet long, laid on 2 by 10 inch sills	Development work	Gravity	2 to 3	(?)	Bonus based on tons per man-shift tramm from stope.
Copper Queen	Seldom necessary	2-inch, 8-foot planks	Development work and old stopes	do.	4	Generally two miners each shift do all work	Bonus based o ground broke with various allowances.[1]
United Verde	Double 2-inch plank on 4 by 12 inch sills 5 feet 4 inches center to center	2-inch plank, 5 feet 4 inches long. No sills?	Open-pit stripping and mine development	Chutes and tramming Cars plus gravity	6 to 7 3	Miners, muckers, timbermen, pluggers fill trammers all different men.	Efficiency ba us or contra cubic-foot ba sis for mine
Frood	Is used	Is used, but details not known	Waste from surface crushing, picking and cobbing plants	Cars	7	(?)	(?)
La Colorada	2 by 10 inch plank, 10 feet long, nailed to 4 by 10 inch sills	3 by 10 or 3 by 12 inch plank	One-fourth from development; rest from surface glory holes	Gravity and scrapers in rills; scrapers and hand in HCF	3 to 6 in rills; 3 in HCF	Two muckers, two miners each shift	Large part c tracted for labor and e plosives on cubic-meter basis.
Pilares	(?)	In HCF 3-inch plank, for high-grade ore. In rills 3-inch plank or round poles for high-grade ore only	Chiefly from surface glory holes; some from old stopes and development work	Cement buggies, cars, or scrapers; gravity in rills	4 in HCF; close as possible in rills	(?)	Contract bra ing, muckin and timberi on cubic-m basis.
Magma	Double 2-inch plank	(?)	Surface glory hole and development work	Gravity	4(?)	Two miners do all work	Contracted cubic-foot sis.
United Eastern	Double floor of 2 by 12 inch by 6-foot plank	Single plank floor?	Development, stope raises and surface mill holes	Scrapers and wheelbarrows	3 to 4	(?)	(?)

1 - Since changed to contract system based on cubic feet broken.

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Table 3. Comparative data on mining details at mines using the cut-and-fill stoping method - Continued

(Section D Continued)

Name of mine	Flooring		Filling			Stope crews	Methods of payment
	Sill flooring	Regular flooring (temporary)	Source of filling material	Method of spreading fill in stope	Filled how close to back, feet		
	24	25	26	27	28	29	30
ampion	None used	None used	Sorting in stopes; development work	Gravity	2 or 3	One miner, one or two muckers do all work in stope	Bonus system based on tons broken, tons sorted, width of stope, and explosives used.
tte district	(?)	(?)	Development work; prospect drifts into walls from stopes; surface glory holes	By hand in HCF; gravity in rills	3 to 5	(?)	Generally contracted on cubic-foot basis.
cs	(?)	2-inch plank	Development work; prospect drifts off stopes, and sorting	Hand shoveling	(?)	(?)	Chiefly day wages.
Utah	(?)	Only with coarse fill, 3 by 10 inch plank, 8 feet long, overlapped 8 inches	Chiefly from development work	Hand shoveling and scrapers	5	Three or four men in a crew do all work in stope; may or may not do tramming also	5 to 7 1/2 cents per cubic foot, includes all operations in stope.
chambre	Hardwood slabs and poles on sill	None, probably	Half from deslimed mill tailing, half from development work	Sand is washed in; rock spread by hand, wheelbarrow or scraper	5 to 6	Contractors have charge of crews of day wage workers	Separate contract price for all operations in stopes.
n Cap	Double floor of 3 by 12 inch planks	3 by 12 inch by 5 foot plank	Development work; waste shrinkage stopes; old stopes	Gravity	3	(?)	Day wage.
Lluride istrict	(?)	Boards, canvas, or steel sheets	From sorting or shot from footwall	Hand shoveling	"Working distance" (7 feet)	(?)	Labor and explosives contracted at $8 to $12 per fathom (36 square feet).
by-Five	Is used if floor pillars are ore	2 by 10 inch plank 10 or 12 feet long, laid on 2 by 10 inch sills	Old stopes and development work	Gravity	2 - 3	(?)	Contracted on square yard basis.

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Table 3.- Comparative data on mining details at mines using the cut-and-fill stoping method - Continued

(Section D - Continued)

Name of mine	Flooring		Filling			Stope crews	Methods of payment
	Sill flooring	Regular flooring (temporary)	Source of fill-in material	Method of spreading fill in stope	Filled how close to back, feet		
	24	25	26	27	28	29	30
Ground Hog	Double 2-inch plank floor laid on 10 by 10 inch sills	(?)	Development	Gravity	2 - 3 ?	One miner, one timberman, one helper do all work together	Probably day wage?
Questa	Single 2-inch plank or split lagging over drift sets	Usually none used. Sometimes plank flooring	Sorting or by shooting down walls to necessary stope width	Hand shoveling	5 to 7	Two to four men do all work	Labor and powder contract on basis of tonnage of mill ore produced.
Santa Gertrudis	(?)	3-inch plank	Development and surface glory hole	Wheelbarrows	Prob. 5	(?)	(?)
Lucky Tiger	(?)	Ore blasted onto cowhides in flat-backed stopes. Plank flooring used in rill stopes	Walls blasted after removing ore in narrow veins. Raises driven into walls when veins are wider than about 3 feet, or development waste used	Hand shoveling	(?)	(?)	Some stopes contracted; payments based on cars tramped from stope and grade of ore produced. Drilling in "company" stopes done on contract. based on footage drilled
Zaruma	(?)	(?)	Development work and waste raises in walls	Gravity and hand	Probably 2 - 3	(?)	Contracted on footage-drilled basis.

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Blasting

Gelatin dynamite of 25 to 60 per cent strength is used in stope blasting in most of the mines considered here; 35 and 40 per cent are the commonest strengths. Ammonia or special dynamites are used in a few mines.

Hard massive sulphides and occasionally other hard ores, particularly when mined in wide stopes, are likely to break in large fragments. Secondary drilling and blasting of this coarse material is an important item of expense. Extra care and more explosive can be used in the primary blasting with an over-all saving in cost if by so doing the formation of large bowlders can be materially reduced. In most cases proper spacing of the holes, the use of high-grade explosive and of tamping will keep the formation of bowlders to a minimum.

In 1922 and 1923 tests were conducted by the Bureau of Mines to determine the efficiency of primary blasting methods, with results expressed as follows:[17]

> Crushing rock by means of explosive is usually more economical than breaking it by hand. Therefore, where bowlders are being formed, more explosive should be used in the drill holes while the extra amount does useful work. In other words, if the walls and back of the stope will withstand the extra jar, an additional cartridge or two in a hole may often be used to advantage.
>
> The results given in this paper indicate that the use of stemming decidedly helps to prevent formation of bowlders, does not increase the cost of blasting perceptibly, and lessens appreciably the amount of work needed to break up the bowlders. The tests show that one stick of stemming to a hole is not enough and that at least two should be used.
>
> The use of high-grade gelatin dynamites makes a substantial reduction both in the size and proportion of bowlders formed, and also increases the fines in hard-ore stopes. In addition to their greater shattering power, the higher-grade explosives have more propulsive energy per pound, and probably that also contributes to the improved results. Even with the present difference in price, the tests show that the higher grades of explosive are more economical to use.

At United Verde the proper fragmentation of the heavy, massive sulphide ore is a difficult problem. The trial of stopers and vertical holes has been referred to under the head of methods of breaking ground. Another expedient tried was the use of cordeau fuse and sand spacing. The sticks of dynamite comprising the charge in each hole were separated by sand stemming to distribute the force of the detonation over nearly the full length of the hole. Detonation was accomplished by using cordeau fuse. This method resulted in an improved fragmentation of the ore, but was not used after it was found that equally good results could be obtained by a different arrangement of the holes in the round. However, any use of explosives or any arrangement of holes that would give the desired fine product without any secondary blasting was found to result at the same time in numerous cut-off holes. Some secondary blasting is now regarded as a necessary evil.

No. 6 and 8 caps and fuse are in general use for firing stope rounds.

The choice of a proper explosive is closely related to the problem of primary versus secondary breaking. Combinations of different kinds and amounts of explosives with different types of drill rounds should be studied to obtain the best results. Better breaking has been obtained in a few instances by using higher-strength explosives for blasting heavy, blocky

17 - Gardner, E. D., and Howell, S. P., Blasting to Lessen Bowlders in Hard-Ore Stopes: Tech. Paper 383, Bureau of Mines, 1926, pp. 22-23.

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ores. Doubtless in some cases explosives are in use which are stronger than necessary.

Under some conditions the prevention of bowlders is particularly difficult. In La Colorada mine of the Cananea Consolidated Copper Co. the ore in the horizontal cut-and-fill stopes broke down in large blocks. It was believed that the jar of the first shot often brought down too-large masses of the ore, thus cutting off or reducing the effectiveness of succeeding holes. To remedy this, small but solid bulkheads of 10 by 10 inch timbers laid flush on top of each other were built up and wedged tightly against the back just in advance of the face to be broken. The bulkheads were thought to prevent the collapse of the back in the form of large slabs at the first shot, thus giving succeeding shots an opportunity to break the slabs. This practice was abandoned upon the introduction of "space blasting," which has resulted in better fragmentation of the ore. Space blasting as used at this mine consists of spacing the explosive throughout the length of the drill hole, instead of concentrating it in the bottom, as is usually done. The sticks of dynamite are separated by wooden spacers 1 by 1 inch square by 10 inches long.

Stemming is not used for stope blasting in the majority of mines discussed in this paper. At United Verde, stemming is used in hard ground. It is used commonly at Cananea and Pilares and in all cases in the Copper Queen mines. At the other mines it is probably not used at all. It has the advantage of lessening the number of bowlders produced in hard ground. Gardner[18] states: "The use of stemming decidedly helps to prevent formation of bowlders, does not increase the cost of blasting perceptibly.....The tests show that one stick of stemming to a hole is not enough and that at least two should be used." To the miner stemming at first seems a nuisance, though it requires little time and effort. Consequently, a decided effort on the part of the management is usually necessary to bring it into use. If the use of stemming is not required to reduce bowlders, it will almost always result at least in some saving in powder, and less poisonous gas is produced when it is used.[19] Against these factors must be noted the possible difficulty of firing missed holes when stemming has been used.

Secondary Breaking

The efficiency of secondary blasting is a subject which has received little attention at most mines, with the result that unsuitable drills and far too much powder are frequently used.

Secondary breaking is done by drilling and blasting, by hammering or by means of a concrete breaker. Its importance varies greatly. In some mines it is estimated that as much as one-quarter or one-third of all the ore has to be broken by some such means following the primary blasting. It is a very expensive process, involving labor and continual delays to other operations entirely out of proportion to the useful work accomplished. Secondary blasting can sometimes be reduced in amount by changes in primary blasting methods, as mentioned previously. Its cost can be lowered slightly by using low-grade explosive, as is done at United Verde, where 35 per cent strength dynamite is used for this work instead of the 50 per cent used for primary blasting. In some circumstances paving breakers can be applied, especially at the grizzlies when scrapers are used to move the ore to that point. These machines, however, are heavy and cumbersome, and might be of little value if the ore were of the type described as "tough." Plugging consists of drilling a 6 to 12 inch hole in a bowlder and blasting with a half stick or more of dynamite. Bulldozing, or breaking by

18 — See footnote 17.

19 — Gardner, E. D., Howell, S. P., and Jones, G. W., Gases from Blasting in Tunnels and Metal-Mine Drifts: Bull. 287, Bureau of Mines, 1927, p. 93.

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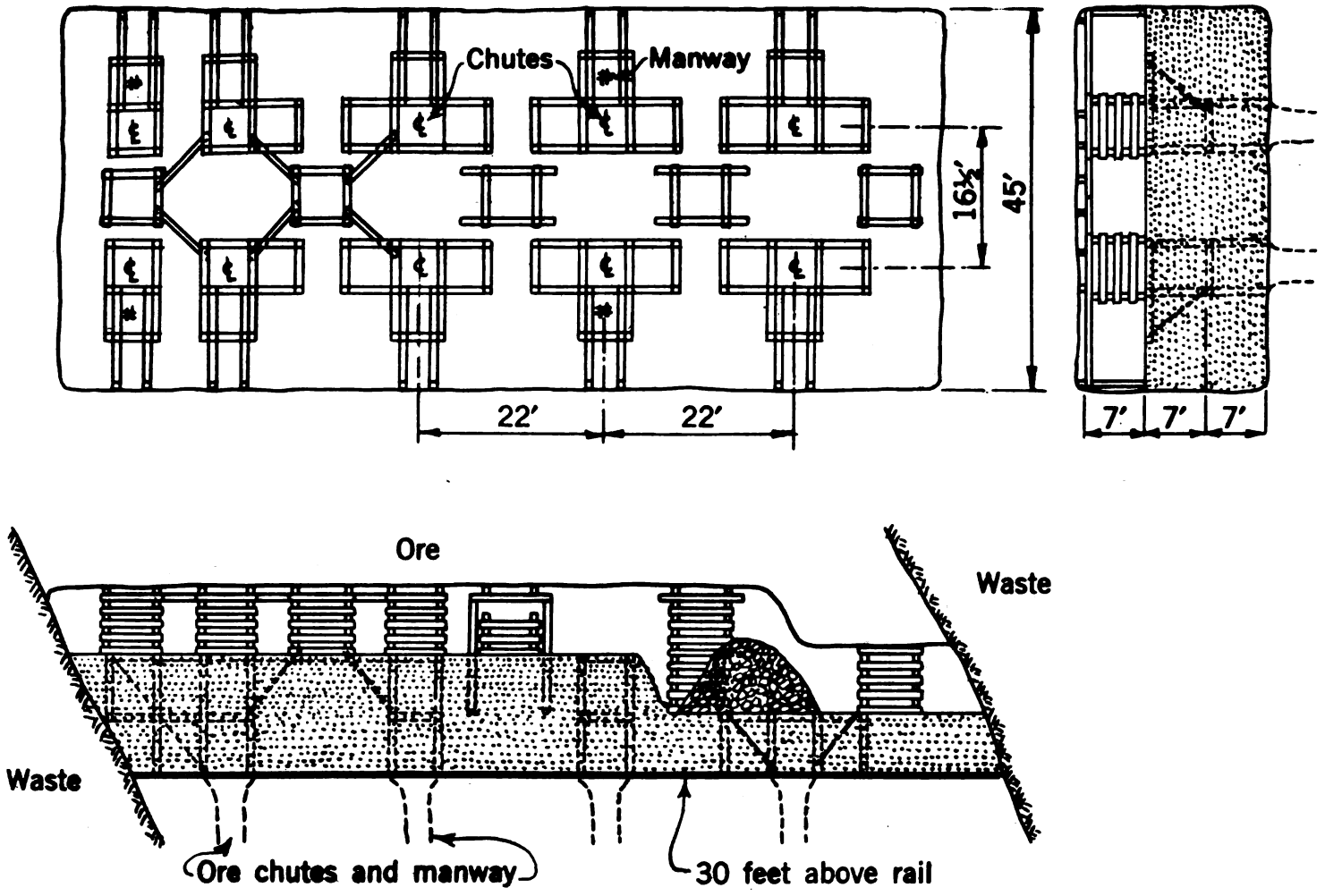

Figure 9.—Timbering to support weak back in cut-and-fill stope at Frood mine

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means of dynamite laid on top of a bowlder, is generally condemned. It saves drilling labor, but is uncertain in action, wasteful of explosives, and frequently dangerous because of the concussion which not only prevents the firing of more than one shot at a time but stirs up a great deal of dust and may damage timber. As secondary blasting can be done only when men are out of the stope, the accumulation of bowlders may delay shoveling or scraping.

A subject closely related to secondary blasting is that of the size of grizzly openings. A full discussion of this would involve many matters foreign to this paper, such as chute construction, haulage, hoisting, and primary crushing. It may be pointed out, however, that in many mines where other methods are used, such as the undercut block caving method, or surface or underground glory holing, a study of the cost of secondary breaking has led to the adoption throughout the mine of larger, heavier, and more rugged equipment, in conjunction with grizzly openings as large as 18 inches. An immediate result of increasing the size of grizzly openings would be an increased cost of maintenance of timbered ore passes and chutes, due to the greater cutting power of large masses of rock. This would be of particular importance in mining a massive sulphide ore. The increased cost would perhaps at some point overbalance the saving in plugging or hammering bowlders. The efficiency of the cut-and-fill method is often restricted by inadequate auxiliary equipment designed in earlier days to handle the product of square-set stopes, or possibly even entirely different types of ores, such as the near-surface oxidized ores.

Temporary Support of Back

There are two general methods of temporarily supporting weak portions of the back of a cut-and-fill stope. One is the use of simple stulls, with or without headblocks or "umbrellas." The other is the building of cribbed bulkheads. Sometimes the use of timber for back support may become so excessive that the cost of mining is increased greatly, both directly through timber and timbering labor costs, and indirectly through the decreased efficiency of other work in the stope. Sometimes the stulls and cribbing used for temporary support can be recovered, to be used again for the same purpose. At Pilares a strong jack with chains is used to pull stulls out of the fill when their usefulness is over. At the Frood mine of the International Nickel Co. of Canada, Sudbury, the backs of many stopes require close support. Cribs of 8 by 10 inch timber 7 feet long are spaced at 16½-foot intervals along the center line of the stopes, which are 45 feet wide (fig. 9).[20] These bulkheads are placed on the sill floor as soon as the first 7-foot cut is taken out, and are extended to the back after each succeeding 7-foot cut. They are taken out before filling, so that the consumption of timber in this process is not as high as would appear at first glance. The ground between cribs is supported by lagging placed on long stringers reaching from crib to crib. Over each chute, in addition, cribs are built up to the back, and long stringers extended from these points to the side lines of the stope.

Timber consumption in a cut-and-fill stope may equal the amount that would be required by square-setting. This was the case in a few stopes at the Pecos mine, yet it was believed that the saving in labor cost resulting from the employment of less skilled workmen than would have been needed for square-setting justified the retention of the cut-and-fill method. The authors are of the opinion, however, that such an excessive use of timber in a cut-and-fill stope should lead to a change of method. When compared with an "open" stope obstructed by a forest of stulls and cribs, square-setting offers enough advantages, such as safety, convenience and systematic production, to more than compensate for the cost of more skilled

20 - Metz, H. J., Mining the Frood Orebody at Depth: Eng. Min. Jour., vol. 130, Nov. 10, 1930, pp. 445-452.

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labor. (See also Gardner, E. D., and Vanderburg, W. O., The Square-Set System of Mining: U. S. Bureau of Mines Information Circular, in press).

Sorting

The horizontal cut-and-fill method is well adapted to mining an ore which requires sorting. Some is done in the stopes at most of the mines noted in this paper. The amount of waste sorted ranges from as little as 2 per cent of all material broken at the United Verde Mine to almost 50 per cent at the mines on the Copper Range.

Sorting in stopes increases the stoping costs per ton of ore by slowing down production; it is not conducive to high rates of production per man-shift. When scrapers are used to move the ore to the chutes, as at Cananea or Park Utah, sorting in the stopes is not feasible except for an occasional bowlder of waste. There are several other conditions which may be adverse to underground sorting in cut-and-fill stopes: High labor cost; low transportation and concentrating costs; an abundance of cheap filling material otherwise available; or simply the character of the ore, as when the distinction between ore and waste is based on an arbitrary assay value and can not be made by inspection.

On the other hand, sorting has many advantages over the practice of sending out of the stope all material broken. By producing a richer ore, transportation and milling costs per unit of metal are reduced, and the required plant capacity for a given metal production is likewise decreased. The maintenance cost of ore passes and chutes is lessened also. Filling is provided in the stopes and in some cases more cheaply than can be done in any other way. If sorting is physically possible, estimates should be made of the importance of the above advantages and disadvantages from a cost standpoint, and operating tests made if the estimates show possible gains by a change in practice.

At the Ground Hog mine, where most of the ore is hauled some distance by truck and treated in a custom mill, not only is a considerable amount of waste removed by sorting, both in the stopes and at the ore bins, but a high-grade, direct-smelting lead ore is produced by sorting out occasional pieces of pure galena and storing them until enough accumulates to make a shipment. It is estimated that about 3 per cent of the rock broken is sorted in the stopes as waste and about 5 per cent more at the ore bins.

At Pilares waste is sorted in stopes where mining conditions permit; it is estimated that 2,000 to 3,000 tons of waste per month is thus sorted and left in the mine. This would be 3 to 4 per cent of the total mine production; as sorting is done only in certain stopes, the proportion removed in these stopes must be considerably higher.

At the mines of the Copper Range Co., Michigan, much attention has been paid to the subject of sorting waste, and the articles referred to in Table 1 contain very interesting discussions of the economics of sorting. The lode in this district is wide (16 to 20 feet) compared with those in the other copper districts farther north. It dips about 70° and has a very weak hanging wall. Schacht[21] gives figures for three years in the Champion mine (probably about 1919 to 1922), according to which, for every 100 tons of material broken in a stope, including ore and waste broken from the lode proper, waste caved from the hanging wall, and waste from footwall exploration, 72 tons was picked over by hand, producing 46 tons of ore (copper rock) and 26 tons of waste. This left 28 tons of waste which, if possible, was allowed to remain where it fell without handling and served as filling. In addition, from 5 to 10 tons of sand filling or development waste was brought in from outside the stope, as about 60 tons of fill was required for each 100 tons of material broken from the solid. Inspectors watched the grade of ore trammed from the stopes as well as the waste fill left in the stopes.

21 — Schacht, W. H., Mining Methods of the Copper Range Co.: Trans. Am. Inst. Min. Eng., vol. 72, 1925, pp. 343–370.

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The new "sublevel" cut-and-fill stoping method in the Champion mine still permits the ore to be sorted, even though all work is rill stoping, instead of being chiefly horizontal work, as formerly. (See figs. 1,K and 1,L.) In fact, the tonnage hoisted represents only about 60 per cent of that broken. The stopes are now worked in sections of relatively low lift, three rill stopes to each 100-foot level interval. As the ore is blasted it falls onto the waste fill, the surface of which lies at the angle of repose, about 38° from the horizontal. There the newly broken material is carefully picked over by hand. Rock showing no sign of native copper is thrown to one side or the other of the rill. Copper rock is thrown down to the toe of the rill for transportation to the chute. This is less difficult than might appear, as the distance the rock would have to be thrown can never exceed 40 feet, and will usually be more nearly half that amount. All fine material is shipped as copper rock, unless the boss orders that it be left in the stope. Inspectors check the cars trammed to prevent shipping poor rock, and the bosses and mine captain watch the discard for copper rock left in the stopes. Mendelsohn states that one very important feature of this stoping method is that, because of the ease of disposing of waste rock, the miners can be encouraged to break the stope wide enough to be sure no copper is missed in the hanging or footwall. Moreover, the transportation of millions of tons of barren or unprofitable rock has been avoided by this practice, and the total saving in cost in 1930 amounted to $0.95 per ton of sorted material left in the mine. This saving in cost was distributed as follows, per ton of waste left underground:

Haulage and hoisting (exclusive of fixed maintenance charges)	$0.25
Preliminary crushing in rock house	.06
Transportation to stamp mill	.24
Stamping	.40
	$0.95

Successful sorting depends to a large extent on the method of payment of the miners, as will be pointed out under "Methods of payment." At the United Verde Mine, the miners are paid a certain amount per ton for all waste sorted and piled in measurable heaps in the stopes The endeavor is to reject any material containing less than 1 per cent of copper. It is stated that this is readily done in the stopes producing schist or porphyry ore, but that it is very difficult in stopes in massive sulphide ore. In the latter, sorting is therefore confined to taking out the material resulting from the presence of narrow, barren andesite dikes which cut the ore. In the four months of January, April, July, and October, 1929, in which a total of 108,479 tons was mined in cut-and-fill stopes, 1,944 tons of waste was sorted out, or 1.79 per cent.

Moving Ore in Stopes

In horizontal cut-and-fill stopes the broken ore may be shoveled directly into the chutes by hand, loaded into wheelbarrows, "buggies," or cars which are then dumped into chutes, or dragged into the chutes by scrapers. Hand shoveling is the method in common use.

A nearly universal practice in horizontal stopes and a common one in rill stopes is to lay a floor of plank on top of the fill before blasting, for which reason the shoveling is less difficult than it would be in an open stope where the shoveling is done on a rock bottom. However, hand shoveling remains one of the most costly single operations in the entire process of mining, and at some mines it is being supplanted by mechanical methods. At United Verde, where shoveling is admittedly difficult on account of the weight and the coarse-

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breaking nature of the ore, it is estimated that the muckers in the stopes handle from 15 to 25 tons per shift, averaging 20. If a mucker earns $4 to $5 per day, this means a cost of 20 to 25 cents per ton. If chutes are more than 25 feet apart it may or may not be advantageous to use wheelbarrows or cars, depending on the weight of the ore, the distances to be shoveled or trammed, and the cost of labor as compared with the cost of installing cars and track or wheelbarrows. Wheelbarrows are used occasionally at Cananea, if it is inconvenient to install scrapers. At Matahambre wheelbarrows are used when a stope widens or lengthens so that shoveling direct into chutes is no longer economical. In neither of these mines, however, is the use of wheelbarrows a part of the stoping scheme as laid out. At the Ground Hog mine, when horizontal cut-and-fill stopes were used, a considerable part of the ore was moved to the chutes by wheelbarrows. This has been eliminated by a change to the rill method of stoping. At Santa Gertrudis, wheelbarrows are used for both ore and waste fill. The floor of the stopes is kept on a 2 per cent down grade toward the chutes to facilitate wheeling the loaded wheelbarrows in that direction. In the earlier type of horizontal cut-and-fill stopes of the Copper Range mines, most of the ore was shoveled directly into the chutes, or mill holes, but when the distance became too great for this, wheelbarrows or cars were used. The car used, which was designed for this particular service, was of 1-ton capacity, with a shallow body open at one end. It was mounted on a turntable so that it could be dumped in any direction. The low height and open end facilitated shoveling into it, and it could be dismantled into parts small enough to be taken through the mill holes into the stopes without undue trouble.

In the new stoping method now in use at the Champion mine, the ore broken in driving the sublevel drifts (see fig. 1,L), which are in reality the sills of the stopes, is pulled to the raise by scrapers. That broken in the course of stoping operations, as noted under "Sorting," is thrown or shoveled by hand to the foot of the rill. When this method of stoping was first adopted the ore was loaded by hand shoveling into small cars and then trammed to the raise and dumped. The cars used were presumably the type described above, and the tramming distance should average less than 100 feet. Later on experiments with the scraper for handling stope ore proved successful and showed savings as compared to the use of cars, even when hand sorting was necessary.

At the United Verde shaking trays were tried for transporting the broken ore from the stope faces to the chutes. These are said to have been satisfactory in operation, and to have indicated that fewer chutes would have been needed than are now used in a given area. However, certain disadvantages prohibited their adoption. It was practically impossible to protect the trays and their mechanism from the flying bowlders of hard, massive sulphide ore, and they were too heavy to be moved from the stope before each blast. Moreover, the saving in numbers of chutes probably would have been offset by the necessity of building heavier ones to withstand the severe service resulting from the concentrated tonnage and the high 150-foot lift.

Although in several instances scrapers are used for spreading fill in horizontal stopes, at only two mines, Cananea and Park Utah, are they used for moving the broken ore in the stopes. The reason given at United Verde, where they have been tried for this purpose, is that the heavy scrapers required to handle the coarse ore tear up the mucking floor. Additional objections at this mine are that the broken ore contains many bowlders too large to be handled without breaking, that scrapers require a large area to operato efficiently, that it is difficult to keep the stopes clean when using scrapers, and that the output of a scraper is less than that of the regular mucking crew of six men. The objection to the use of scrapers at many mines is the impracticability of sorting waste. Figures 10, 11, and 12 show the scrapers used at La Colorada mine, Cananea, and at Park Utah, and the haul-back sheave designed at the latter place to eliminate the many difficulties experienced with the

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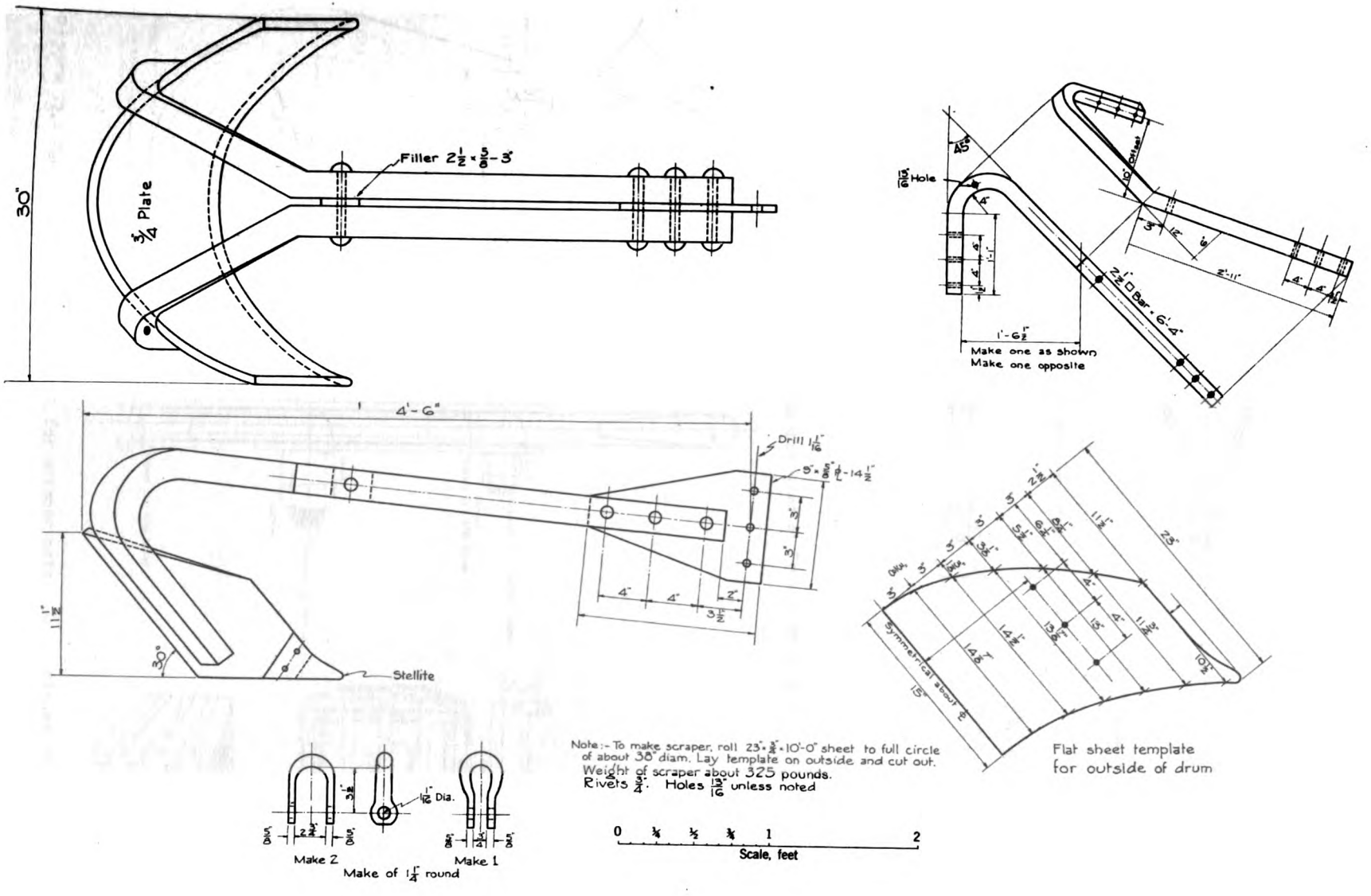

Figure 10.—Crescent scraper used at La Colorada mine, Cananea

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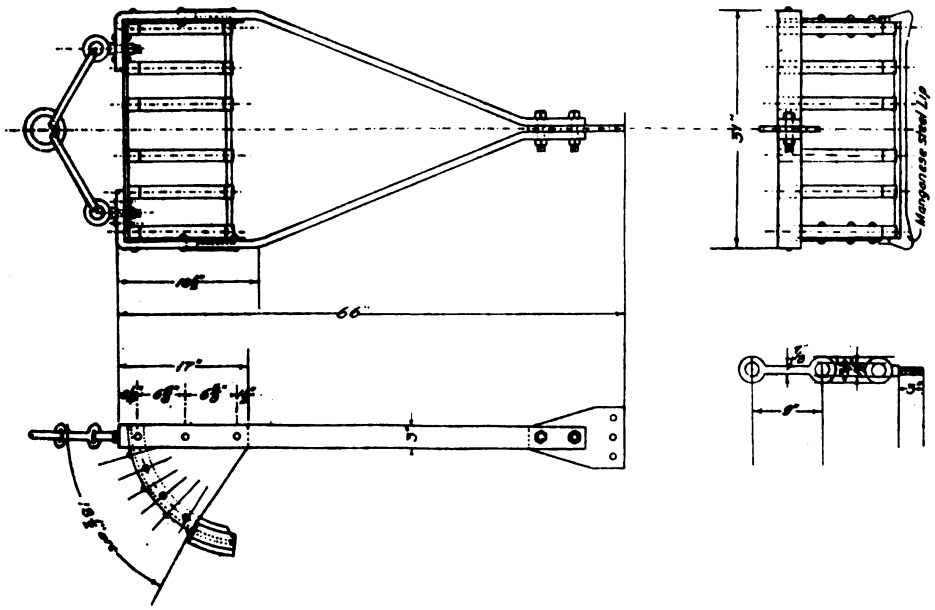

Figure 11.- Scraper used at Park Utah mine

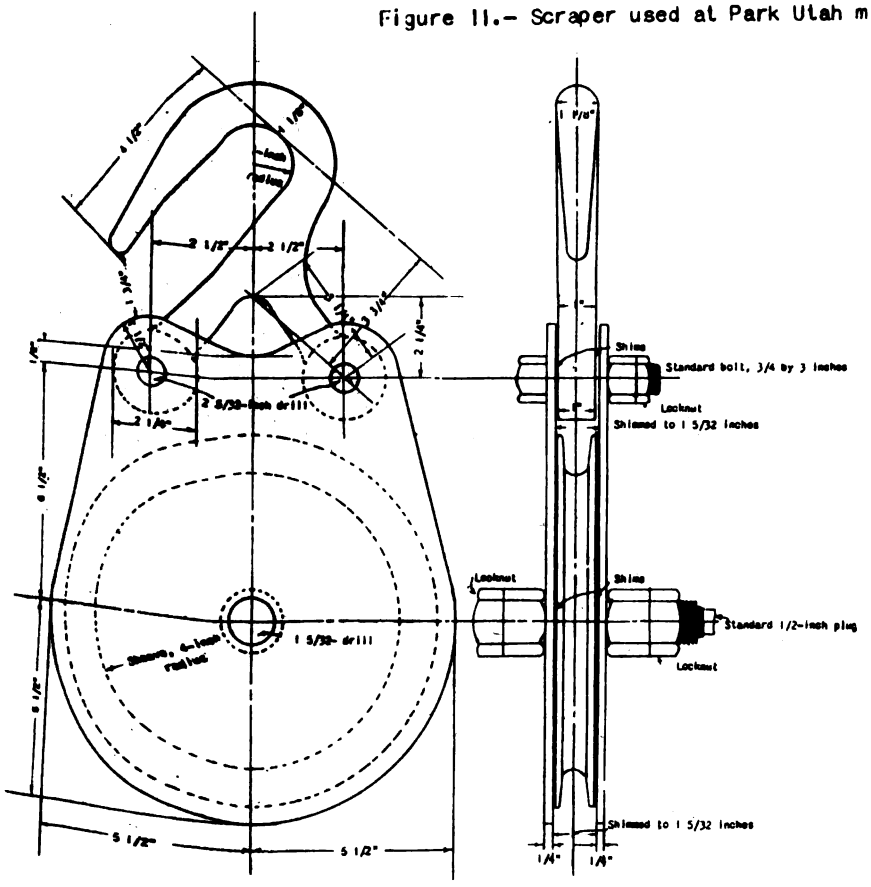

Figure 12.- Park-Utah-type sheave block for scraper hoist

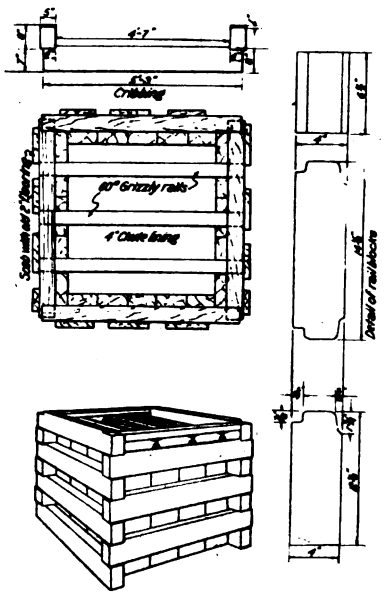

Figure 13.- Single-compartment cribbed chute used at United Verde mine

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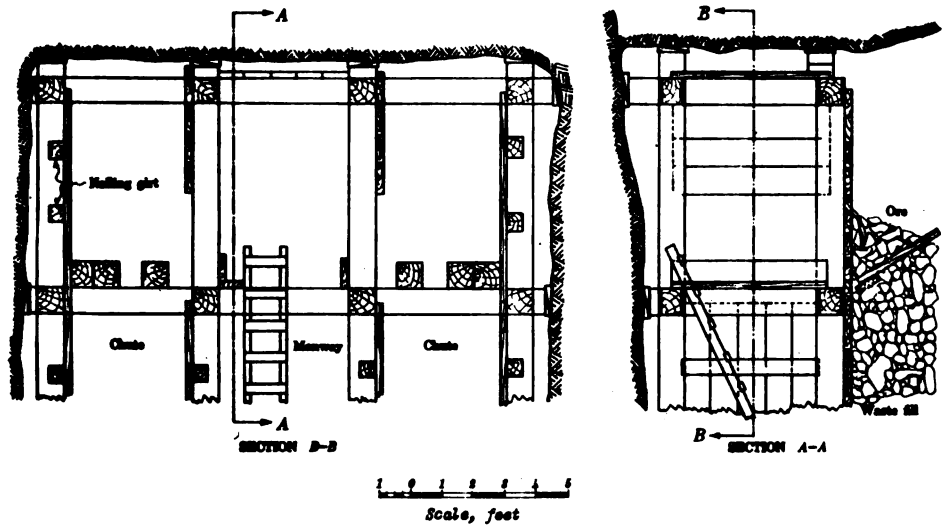

Figure 14.- Square-set ore chutes and manway used in rill cut-and-fill stopes at Magma mine

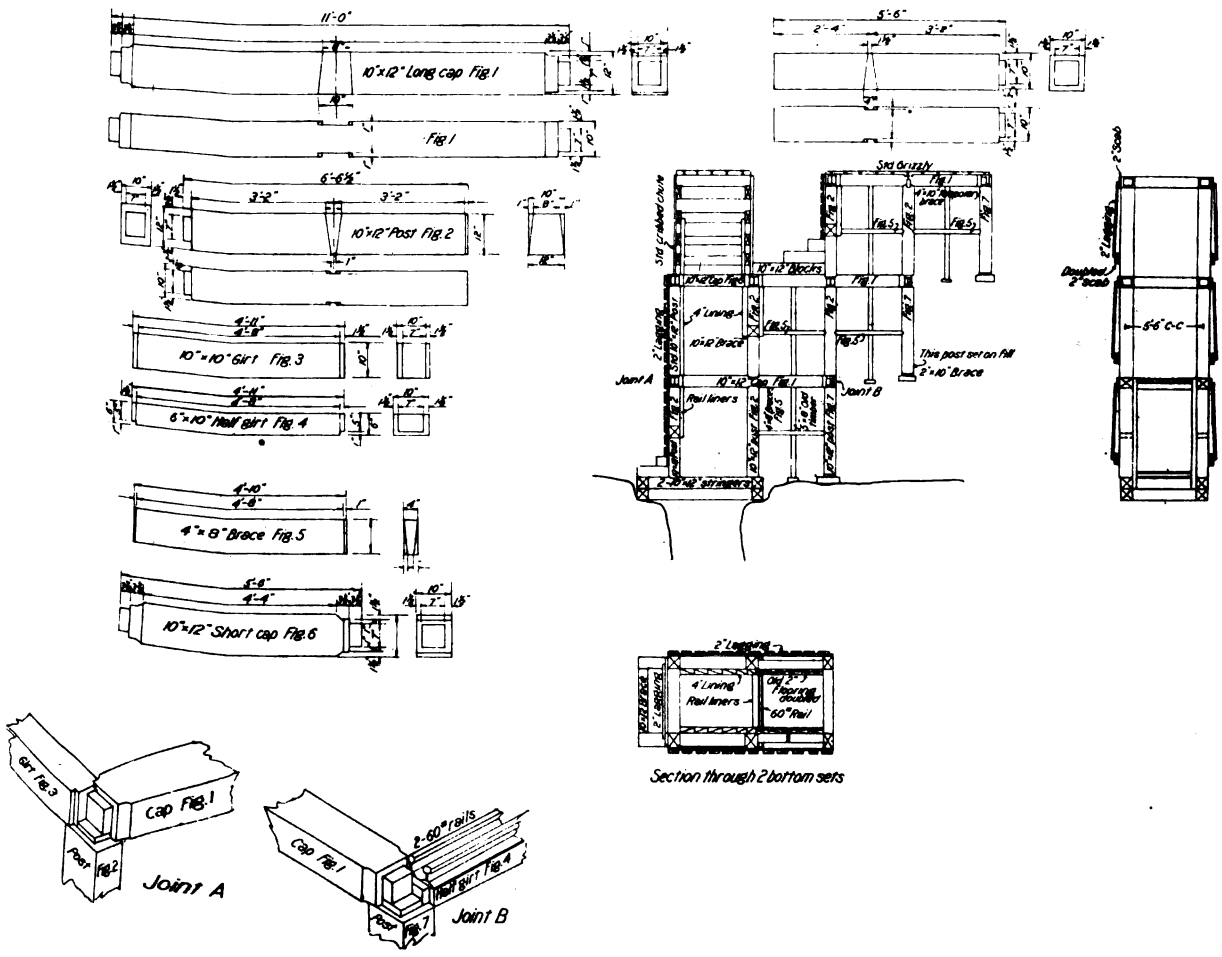

Figure 15.- United Verde 6-post offset chute, using square-sets

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ordinary types of snatch block. Heavy scraper equipment is advisable for coarse or heavy ore. The scraper shown in Figure 10 weighs about 325 pounds. The usual type of hoist for this service is the double-drum compressed-air hoist of 10 or 15 hp. Larger hoists are sometimes used, and the electric hoists recently developed are proving very satisfactory for slusher service. One of the objections to scraper work has been the necessity of continually changing the position of the sheave in order to cover more than a single narrow path. Three-drum hoists are now on the market which obviate this difficulty to a large extent. Two tail ropes are used with this hoist, running through sheaves placed at opposite sides of the muck pile. The scraper can thus be returned to any desired point within the triangle defined by the hoist and the two sheaves.

In a rill stope the ore moves down to the chute by gravity, with more or less shoveling, depending on the steepness of the rill. At Cananea the stopes are so wide that if the slope of the back had been carried on a steep enough angle to make the ore run freely down the rill, it would have formed a dangerous hanging brow or "belly" directly over the chutes. Therefore, scrapers were used to move the ore down to the grizzlies.

In either form of stope little or none of the ore will run freely into the chutes, because of the necessity of having grizzlies over the chutes to prevent large bowlders from plugging them. Therefore, some labor is involved in picking or shoveling the ore from the toe of the slope into the chute; and breaking any bowlders which are too large to pass the grizzly.

Ore Chutes and Manways

Ore chutes are usually constructed of cribbing (see fig. 13). The cribbing is generally lined with plank, which can be replaced when worn through. However, at Cananea it was found that by making the lower portion of stope chutes, which received the most wear, of 10 by 10 inch cribbing instead of the standard, 4 by 10, lining could be omitted, and still the chute would last out the life of the stope. In this way the delay and expense of repair work on chute lining were eliminated and the timber consumption was little if any more than formerly. Under most conditions cribbed chutes are more convenient for cut-and-fill stopes than square-set chutes. They can be built to any desired height by adding the required number of rounds of cribbing. They do not have to resist great pressure from the fill, or any appreciable movement, such as would take place in a shrinkage stope. Sometimes, for additional rigidity and protection against blasting before being covered by fill, cribbed chutes are slabbed on the outside with lagging, scrap flooring, or other waste timber.

Several mines use square-set timbering for extraction chutes. At the Frood mine the square-set chutes are built up one set above the shoveling floor and cribbing placed on top of them to support the back. One or more sides of the set are left open to receive the ore. The manways in these same stopes are cribbed. The chutes in the Magma mine, where the rill method is used, are likewise of square-set construction. (fig. 14). The square-set chutes support the roof and at the same time protect the shovelers from bowlders rolling down the incline.

In narrow veins stulled raises are often used, with either round or square stulls cut to reach from wall to wall at about 5-foot vertical intervals, and lined with plank to form chutes and manways.

At United Verde, where the ore is extremely hard and abrasive and the extraction chutes have to be offset or branched frequently to accommodate the changing floor plan of the stope as successive cuts are taken, elaborate and substantial methods of timbering have been developed and standardized for extraction chutes and manways. One such chute is illustrated in Figure 15.

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Chute construction and repair are often important items in the cost of cut-and-fill mining. At United Verde it is estimated[22] that one timberman and a helper are required to construct the chute raises, manways, and waste chutes in a stope producing about 150 tons per day. Frequently an equal amount of labor is needed each month to keep the chutes and manways in repair. The 1929 labor cost for timbering in horizontal cut-and-fill stopes is given as 20 cents per ton, which is almost equal to the labor cost of breaking the ore. Any improvements in chute building which will reduce construction or repair costs are obviously worth while. Timber consumption for the same period is given as 7 board feet per ton, or a cost of perhaps 35 cents. Conditions are unusually severe at this mine. At some properties, as, for example, United Eastern, the opposite may be true. At that mine chutes were constructed of 3 by 12 inch cribbing laid edge to edge and only about 3 feet square inside.

When it is necessary to offset or branch the ore chutes, the junction or offset point becomes a point of exaggerated wear. The floor can be protected by constructing the chute so that rock builds up at this point enough to cover the timber. The lip over which the ore slides into the main chute can be protected by a rail or heavy angle iron. If the distance from the top is more than a few feet, and the ore falls freely, it will rebound against the opposite side of the main branch with such force that it soon cuts out the cribbing or lining on that side. One remedy is to line that side of the chute with rails. At Cananea the practice is to hang a curtain of short rails in the center of the raise, which checks the speed of the ore and prevents damage to the chute.

Spacing of Chutes

The spacing of extraction chutes varies greatly, 25 feet being representative of typical horizontal cut-and-fill practise and 150 feet of rill stopes laid out along the vein. When a rill stope is laid out across the vein, the intervals between chutes are equal to the width of the stope, as 30 feet at Magma. In horizontal stopes, spacings of 16½ feet (three sets in the haulage drift) in United Verde massive sulphide stopes and in the Frood mine, and 165 feet at Santa Gertrudis, are noted as the minimum and maximum distances. It is stated that the close spacing of chutes at United Verde is due not only to the weight of the ore, which makes shoveling costly, but to its abrasive nature which, if the spacing is too great, causes chutes to wear out before the stope is completed. In this same mine in the stopes working on siliceous ore the chute spacing is 22 feet. The greater distance not only gives more room to store the waste which is sorted out from this type of ore, but encourages sorting, as the bulk of the ore has to be handled rather than rolled and barred down into the chutes.

At the Santa Gertrudis mines the ore is shoveled into small cars and trammed to the chutes, and the long tram may be cheaper than building and maintaining chutes at closer intervals. At Park Utah, where scraping is used, chute intervals range from 50 to 150 feet.

In rill stopes it is customary to have only a single extraction chute at the foot of each rill; in stopes along the vein each chute generally serves two rills. At Cananea, where scrapers are used in the rill stopes, as mentioned above, the chutes contain three compartments and are usually cribbed: the scraper hoist is set up over the central compartment. By turning the hoist through a half circle it is made to serve both slopes of the rill from the one set-up.

At many mines, during silling-out operations for rill stopes, the common practice is to use one or more temporary chutes between the ends and center of the stope to reduce the

2 – Cowley, J. F., and Quayle, T. W., Development and Mining Methods: Min. Cong. Jour., vol. 16, April 1930 (United Verde number), p. 319.

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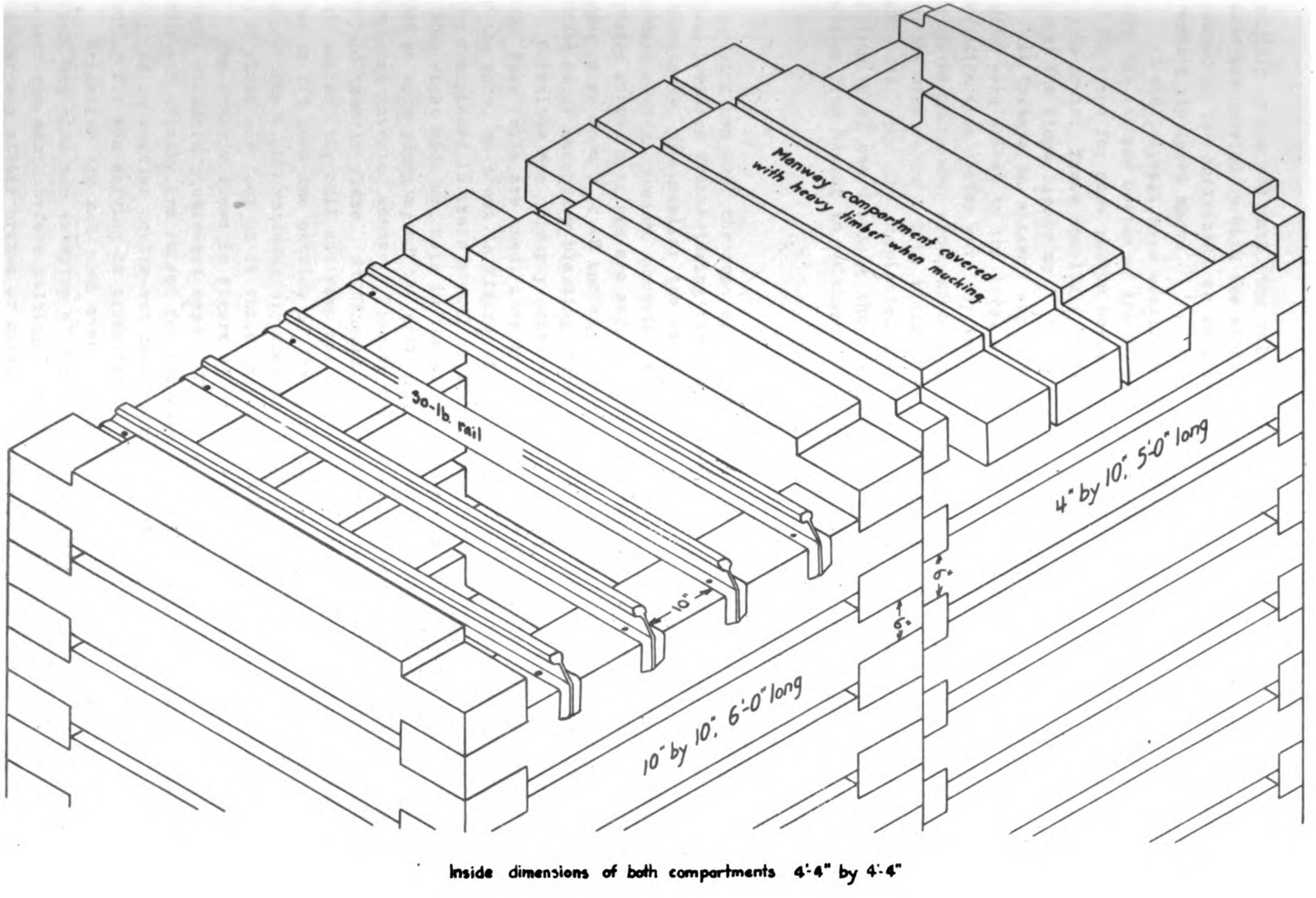

Figure 16.—Type of rail grizzly used on ore chutes at Cananea

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handling of the ore until the rill is fully established. At Pilares one of these chutes is sometimes carried up with the stope and is found to be of great value if the stope has to be changed to the horizontal or square-set method because of ground conditions or because of reaching the level above.

At the Campbell mine auxiliary chutes are sometimes carried up in the fill, midway between the top and bottom of the slope. These are found to reduce greatly the shoveling labor if the stope for some reason becomes so flat that the ore will not all run by gravity to the bottom chute. These auxiliary chutes start near the footwall, but swing out toward the center of the stope higher up.

At Cananea an attempt was made to subdivide the long rills into two sections. Fill raises were driven to the level above halfway between the upper and lower ends of the rill, and extraction chutes were carried up through the fill at the same point. The intention was to secure continuous production by mining the upper half while the lower half was being filled, and vice versa. This would also have partly evened out the very irregular demands for waste fill upon the completion of the large cuts. The idea was abandoned because of the difficulty of coordinating the two sections of the stope and because of the complications arising from having to bulkhead one part of the stope off from the other.

Grizzlies

Grizzlies over the tops of ore chutes in stopes serve the double purpose of preventing large bowlders from entering the chutes and of preventing men from falling into the chutes. Undoubtedly, the smaller the openings, the greater will be the efficiency of handling in loading chutes, trains, pockets, skips, and ore bins. The safety factor is likewise increased as the openings are made smaller. The spacings noted in the mines discussed in this paper range from 9 to 12 inches. The question of the size of grizzly openings has been mentioned under secondary blasting, from the viewpoint of production efficiency.

Grizzlies are commonly constructed of rail or of square, framed timbers (figs. 13 and 14). When rails are used, they are generally turned upside down and held by spacing blocks at the ends, as shown in Figure 13. Bowlders tend to become wedged between the rails when they are placed in their normal position, base down. At Cananea, however, grizzly rails are placed right side up, both in the stopes and over shaft pockets. It is claimed at this mine that no more plugging of the grizzlies by bowlders occurs than if the rails were placed in the usual inverted manner. Placing the rails in their normal position makes it much easier to hold them in place. At Cananea, for use in stopes, rails are cut to proper length, the ends heated, the ball and webb cut off for several inches at both ends, and the base turned down to fit over the outside of the chute cribbing (fig. 16). Placing the rail right side up provides a less satisfactory footing, if it becomes necessary for a workman to step across it or place one foot on it while hammering bowlders.

The grizzly shown in Figure 14 has several satisfactory features. The cross bars are simply standard square-set caps. They are used during one cycle of stoping, and when the chute and grizzly are raised for the next cut are taken out and used as stulls, for cribbing, or in regular square-set construction. The wide, flat surfaces offer relatively good footing for men working on grizzlies.

Grizzlies are not used over the chutes in all mines. The Champion mine and the Ground Hog mine are examples of this. In these mines the ore is all worked over carefully to sort out waste before putting it in the chute, in which process bowlders over a foot in diameter are either broken or rejected as waste. The tops of the chutes are partly planked or cribbed over to prevent men from falling in.

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Flooring

Permanent or sill flooring and temporary flooring are used in cut-and-fill stopes. The former is laid when the stope sill has been cleaned of ore and before the first waste fill has been placed in the stope. Its purpose is to make easier the mining of the ore under it, in the extraction of which spiling would ordinarily have to be used if the fill were not supported in this manner. Permanent flooring is, of course, unnecessary if it is known that the ground underneath will not have to be removed. At some mines it is not placed unless the ore under it is of relatively high grade. This is the case at Pilares, where, Leland[23] states, the material used for fill cements so tightly that there is little tendency for the fill to run by the time a stope is mined underneath it. The sill flooring, when used, is generally a single layer of 2 or 3 inch plank, or a double layer of 2-inch plank. The plank floor is most often laid on cross sills of the same size or heavier timber, as 4 by 12 inch (see fig. 17,A). The use of sill flooring will add 1 or 2 cents, or conceivably 5 cents per ton to the cost of stoping. In some cases this is not justified. On the other hand, a run of waste into a square-set stope when removing pillars might conceivably be just as expensive, and hazardous in addition. Flooring should therefore be used until the behavior of the fill material can be studied, and the omission of flooring should be tried where unfavorable results will not be too costly.

The temporary or shoveling floor is laid on top of the waste fill before mining each cut of ore, and it is taken up before running in the next layer of fill. It serves the double purpose of aiding shoveling and preventing the ore from mixing with the waste fill in the stope. The latter function may be very important when the ore is of high grade or where the valuable mineral tends strongly to be concentrated in the fines. At Telluride, in narrow veins, boards, canvas, or steel sheets were sometimes laid down before blasting ore, to prevent loss of fines. This temporary flooring was then taken up when the footwall was blasted down for fill. Incidentally, because such precautions were not always taken, the loss of fines was so great as to cause the abandonment of the cut-and-fill method. It is estimated[24] that as much as 12 to 25 per cent of the valuable ore was lost when no floors were used. In fact many of the old fills were later drawn out as ore, at a fair profit. Cowhides were used at El Tigre to prevent loss of fines in the "gob". In some stopes in the Bunker Hill & Sullivan mine, in veins of the Jersey type, the shoveling floor was sometimes covered with a few inches of fine waste, to prevent the fines going through the cracks in the flooring. This waste was shoveled into the chutes along with the ore. The dilution was less serious than the loss otherwise experienced.[25]

Ordinarily such precautions are not necessary, however, and the shoveling floor consists simply of 2 or 3 inch plank laid carefully on the leveled waste. Sills may or may not be used, depending on the necessity for a strong, tight floor. At the Park Utah mine, the floor is made of 3 by 10 inch plank, 8 feet long. If the fill underneath the floor is fine material, which makes a smooth bed, the planks are simply overlapped 8 inches at the ends and nailed together. If the fill is coarse, cross-sills are used under the ends of the planks to give an even foundation. Breakage of flooring is much reduced by preparing an even surface for it to lie on. At the Campbell mine the floor consists of 2 by 10 or 2 by 12 inch plank, 10 feet long, nailed to sills of the same material. Before taking the floor up, pre-

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23 — Leland, Everard, Mining Methods and Costs at the Pilares Mines, Pilares de Nacozari, Sonora, Mex.: Inf. Cir. 6307, Bureau of Mines, Aug., 1930, 34 pp.

24 - Bell, C. N., Mining Methods of the Telluride District: Trans. Am. Inst. Min. Eng., Vol. 72, 1925. p. 554

25 - Brown, U. E., Mining Methods of the Bunker Hill & Sullivan Mining and Concentrating Co., Kellogg, Idaho: Inf. Circ. 6407, Bureau of Mines, Jan., 1931, p. 3.

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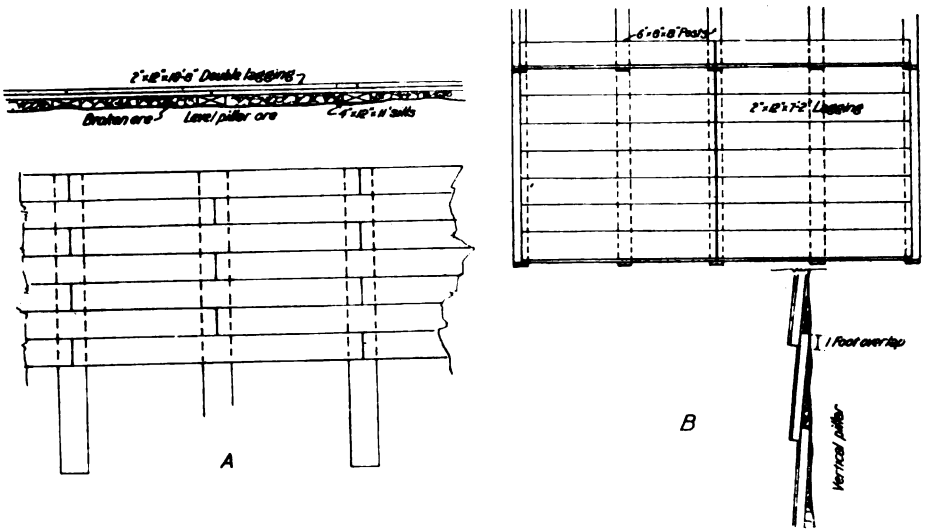

Figure 17.— A, Sill flooring in cut-and-fill stope at United Verde; B, gob fence used at United Verde

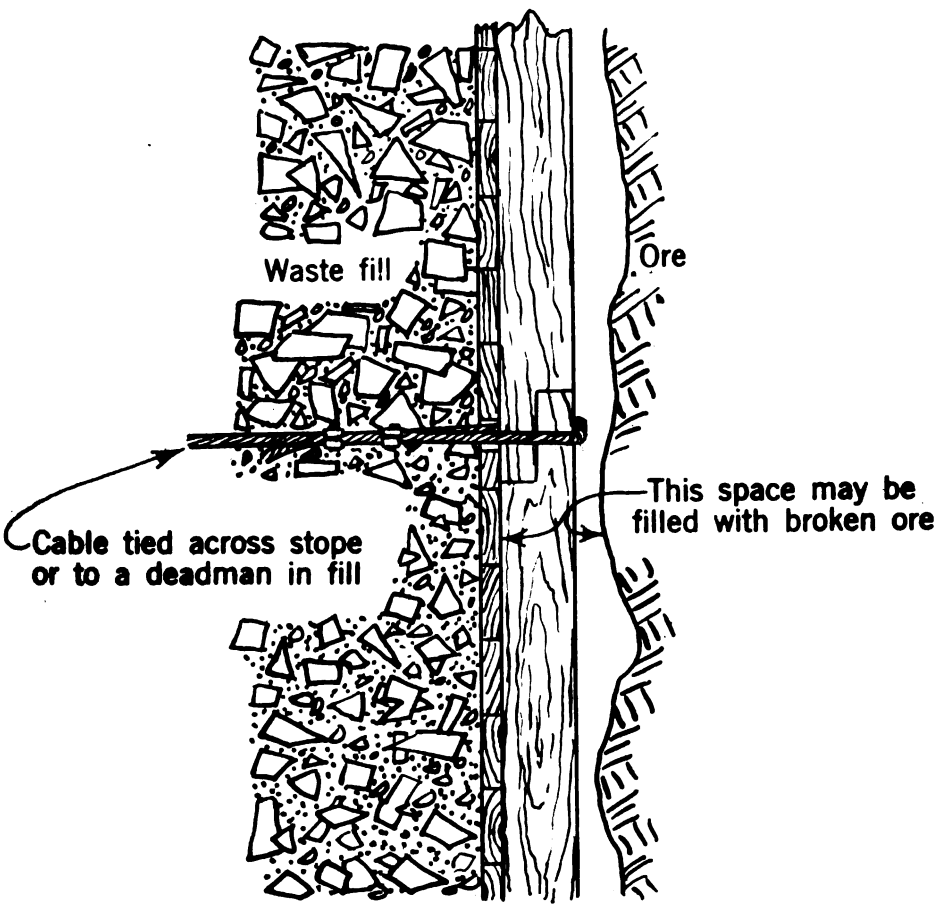

Figure 18.— Gob fence with posts framed in half

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paratory to running in fill, it is carefully swept with heavy brooms. This sweeping is said to yield a considerable quantity of fine ore which has a high copper content.

Flooring adds considerably to the cost of mining. The labor of laying the timber is probably much greater than the cost of the timber used. The latter item, however, might range from ½ to 3 cents per ton, under varying conditions. Assuming that the cost of flooring including labor, might be as much as 8 cents per ton and that copper in the form of broken ore in the stope is worth 2 cents per pound, then the flooring in a stope would have to save from loss in the fill at least 4 pounds per ton of ore, or 5 per cent of the copper in a 4 per cent copper ore, in order to justify its cost on this count. The value of the flooring to the shovelers is, of course, considerable. However, if the grade of ore were to decrease, or the cost of flooring increase, a point might be reached where the ore and the shoveling labor saved would not pay the cost of flooring. At Pilares, temporary flooring is used only in high-grade stopes and even in rill stopes flooring is not always used, because when the fill is thoroughly wet and packed down on the slope little loss of ore occurs. After the completion of a slice the surface of the fill is carefully scraped down to recover fines.

In the rill stopes of the Champion mine no flooring is used, either on the sill or when stoping. Under the present system it is unnecessary to support the fill above an old level when stoping up under it, so that no sill flooring is needed. All ore is picked over by hand on the slope of the rill and large quantities of waste are put aside at that point. Hence no temporary flooring is desirable or even permissible.

In rill stopes when flooring is used it is customary to nail cleats on the floor to make climbing up and down easier.

Gob Lagging

Usually if a cut-and-fill stope is partly bounded by ore which is to be mined in the future, a fence should be erected between the waste fill and this ore. This will prevent runs of fill when mining the adjacent ore. It will also help, when using a stope and pillar system, as at Magma, United Verde, and many other mines, to keep the stope properly aligned. Any space between the gob fence and the solid ore may be filled with broken ore or old scrap timber to support the fence. Often the posts are tied by wire rope or old cable strands to the fence on the opposite side of the stope or to a "deadman" buried in the fill. This prevents bulging of the fence. Figure 18 shows the method of building a fence by framing the posts in half.

Figure 17,B shows the type of gob fence used at United Verde. At this mine horizontal stoppings are put behind the fence at each floor so that when the pillar stopes come up, only one floor of the loose ore stowed behind the fence can run out at one time. Otherwise, considerable runs might occur and undue strain be thrown on the fence.

Some types of stope filling pack so tightly as not to run when mining alongside. If such material is used no advantage is gained by building gob fences.

Filling

It is obvious that in a cut-and-fill stope the fill must be placed in volume nearly equal to the ore removed. The difference would be the space taken up by chutes, timber, and manways. If it is assumed that 10 cubic feet of a sulphide ore constitutes 1 ton, then, roughly, ½ ton of broken limestone waste will be required to fill the space made by taking out the ore.

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If mining and filling are alternating processes, rather than continuous, the demand for waste fill will be intermittent. In large rill stopes, such as those formerly worked in La Colorada mine, several thousand tons may be required to fill the opening made by one mining slice. Ore production from the rill necessarily ceases during the filling period, which makes large and quickly available reserves of broken waste very desirable. If operations can be planned so that the daily waste requirements for the mine as a whole will be nearly constant, the filling cost will be appreciably lower because of the more efficient organization and smaller required capacity of the fill system.

Nature of Filling Material

Broken waste rock of any size that can be economically handled in regular mine cars, passed through chutes and raises, and spread smoothly enough in the stopes to support a floor of plank is usually satisfactory as a stope filling. A paper on iron mining in the district of Krivoy Rog, Russia,[26] contains some interesting notes on the characteristics of the material used for fill. The ore there is very heavy, and the walls apparently need prompt and unyielding support. The conclusion is reached that a mixture of about three parts of wet clay and one part of finely crushed rock, such as quartzite, forms the best filling. This material showed less shrinkage upon packing and drying than dry rock or rock and clay mixed in other proportions. Such care is seldom necessary, however, and the expense of the filling methods described would be prohibitive in the United States. Fine rock is preferred to coarse, because it is easier to transport and spread. It also gives a better surface to support the shoveling floor, with the result that less flooring is broken. Development waste is usually finely broken and is, therefore, sometimes used to form a smooth surface on top of coarser waste fill from other sources. Tailings from a concentrator make very satisfactory filling and are used in a few cases. They can be spread by hand or washed into place as a hydraulic fill, in either case more cheaply than broken rock.

Waste high in sulphur content (pyrite) should not be used for fill, because of the danger of fire. Sulphide ores oxidize rapidly when exposed to air and moisture. The resulting heat and gases would be objectionable in any case, and if timber or combustible waste were present in the fill, even in small amounts, spontaneous combustion might start the fill burning, with serious consequences.

Material containing some earth or decomposed, soft rock, or which for any reason tends to cement or pack together is very desirable for filling, as future mining operations underneath or alongside will be much easier than with material which runs freely. Recently at Cananea a considerable part of the fill used has been composed of an extremely altered and decomposed tuffaceous rock. This packs so tightly in the stopes that when it is used no gob fence is necessary.

Sources of Supply

Generally a mine has more than one source of supply of waste fill. As an example, the Pecos mine uses waste from development work, from prospect drifts out of the stope itself, from sorting in the stope, and formerly also from surface glory holes.

Development operations are the commonest source of filling, however. It is logical to use this material for fill as it is often the cheapest means of disposing of a considerable amount of waste produced in the progress of prospecting and development work. It is very

26 — Bouldovsky, A. K., and Ayvazoglou, W. (translator), The Method of Underground Mining of Iron Ore in the District of Krivoy Rog: Inf. Circ. 6254, Bureau of Mines, March, 1930, 48 pp.

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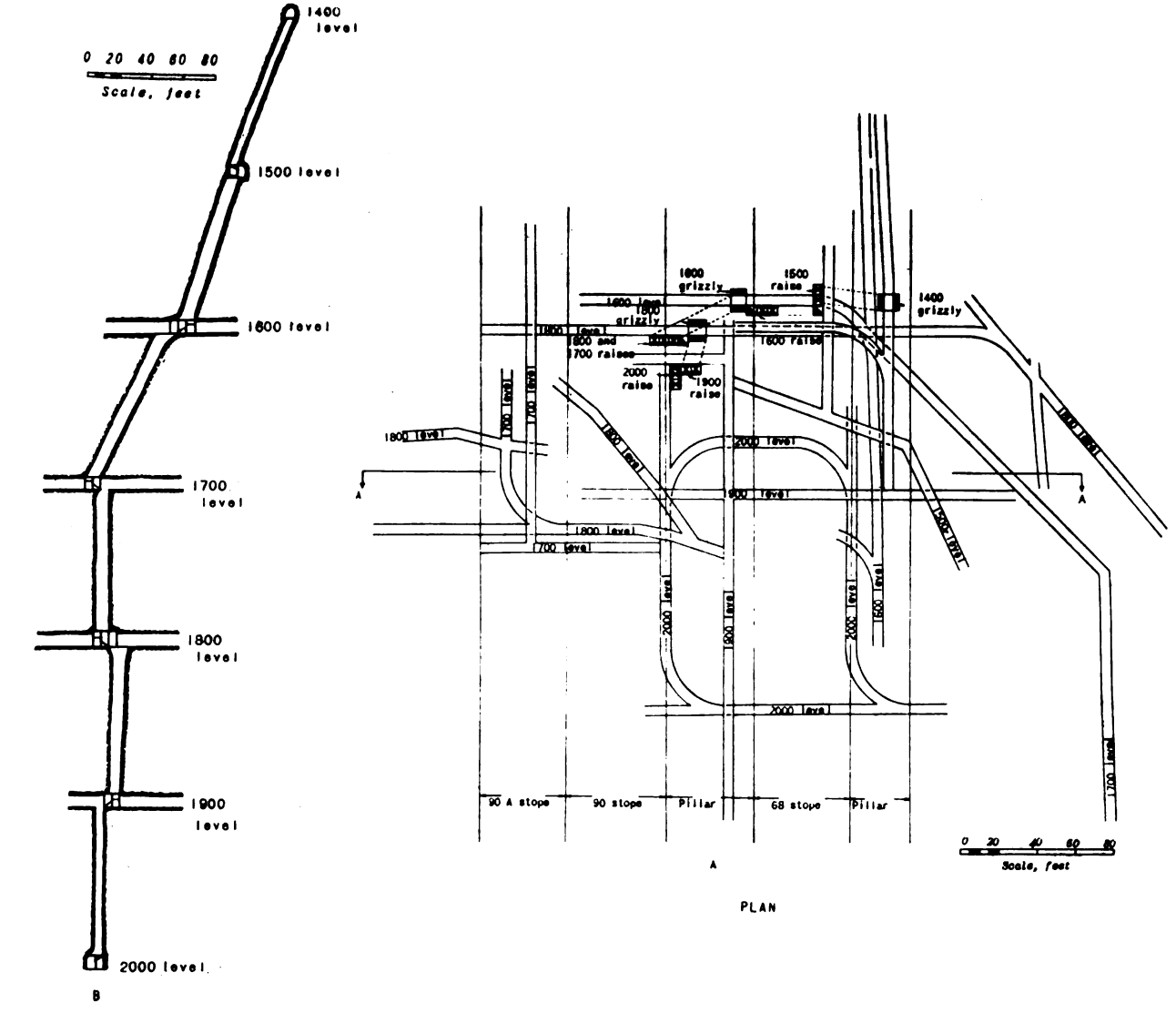

Figure 19.- Waste storage and transfer system, Campbell mine

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rarely, however, that development work can supply fill for all the ore removed, as simple calculations will show. Assuming that a half ton of fill will be required per ton of ore, and that the average development opening produces 50 cubic feet of solid waste, or not over 4 tons per linear foot, the mine would have to maintain a development ratio of at least 1 foot of development work for every 8 tons of ore produced. This is an extremely high ratio, and not often necessary. For instance, Quayle states[27] that at the United Verde it is necessary to drive 20,000 feet of drifts and raises yearly to maintain a daily production of 3,000 tons. This represents a ratio of approximately 45 tons of ore per foot of development. The annual reports of the Magma Copper Co. for 1927 to 1930 show a total of 61,500 feet of development work and a production of 1,000,000 tons of ore, a ratio of 16.4 tons per foot of development. At the Pecos Mine for 1929 the ratio was 16.3. At Pilares, from 1925 to 1929, the figure ranged from 20.8 to 32.9 and averaged 25.2 over the entire period. The Butte mines of the Anaconda Copper Co. in 1926 and 1927 reported an average of 14.2 and 18.0 tons, respectively, produced per foot of development work. The mines of the Bisbee district are less fortunate than most in this respect, their development ratios averaging 9 to 12 tons per foot of development. Another disadvantage of depending on development work to supply all the waste fill for stopes is that the two operations must be balanced, which is perhaps theoretically good practice over long periods, as a year or two, but which is not compatible with the frequent necessity of controlling production rates by shifting men from stoping to development work, or vice versa.

If development work is to supply a large part of the waste fill for stoping purposes, it is desirable to provide large waste storage capacity so that the irregular demands for waste fill may be met. The Calumet & Arizona's Campbell mine has a waste raise and pocket system for this purpose (fig. 19). Waste can be put into the system at the 1400 or any level below and drawn out at any level down to the 2000. Thus hoisting of waste is avoided, there is always room for the disposal of development waste, and a supply of filling material is always on hand.

Glory holes on the surface often are relied on for the major part of the waste fill. If a glory hole can be located directly above the stoping zone in the mine so that the product can be transported chiefly by gravity, the cost of the filling material will be very low. It will seldom cost more than 10 cents per ton of waste passed through a 12-inch grizzly and ready for delivery to cars on the waste haulage level, making a cost of 5 cents per ton of ore stoped, excluding handling and spreading charges. Glory-hole operations comprise a separate and important mining method and will not be discussed here in detail. Briefly, a raise is driven to the surface; at some point in the raise, well below any probable future depth of excavation, a grizzly is installed and a grizzly chamber provided, or bulldozing or bombing or blasting chamber, as it is variously called. Below this point there should be a capacity of several trainloads of rock and a chute for loading the broken rock into cars. At the surface the rock is drilled and blasted into the raise, the excavation taking the form of a funnel, with sides steep enough so that the rock will run by gravity into the raise. Drilling is generally done with hand-held jackhammers, drilling vertical holes 6 to 12 or more feet deep, which are blasted with dynamite without springing. The explosive consumption for this type of work is usually quite low (at Cananea it was 0.12 pound of dynamite per ton of rock broken), because material suitable for filling is of a type that breaks readily due to being naturally strongly fractured or jointed. Therefore, it would seem to be good economy to use sufficient holes and load heavily enough to break the rock well with the primary blasting, thus reducing the labor cost of passing it through the grizzly. Two

27 - Quayle, T. W., Mining Methods and Practices at the United Verde Copper Mine, Jerome, Arizona: Inf. Circ. 6440, Bureau of Mines, February, 1931, p. 8.

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men usually work together at the grizzly, breaking the oversize bowlders either with hammers or by plugging.

At the Magma mine an 8-inch jaw crusher is installed below the grizzly to crush the rock to a size suitable for stope filling. The grizzly consists of 60-pound rails crossed to make openings 16 inches square. A miner and helper do all the work in the glory hole and at the grizzly and crusher, while two other men are employed most of the time in keeping the rock moving through the system of waste passes leading down to the stoping levels. The waste tends to pack in these raises if allowed to stand, especially when wet. To prevent water from the occasional heavy rains from entering the waste raises, a drainage raise taps the waste raise just below the bottom of the glory hole. At this mine the cost of filling (all stopes are carefully filled) has ranged in the last five years, 1926 to 1930, from 8 cents to 20 cents per ton of ore mined, depending on the proportion of fill which was derived from development work and that from the relatively costly glory-hole operation.

At Cananea in the latter part of 1929 about one-half of the waste fill for La Colorada mine or approximately 500 tons per day came from a glory hole on the hillside near the mine. The cost of this material, delivered to the loading chutes below the glory hole was $0.0983 per ton of waste during a typical month, or probably about 5 cents per ton of ore mined.

Since 1929 an increasing proportion of the waste fill for La Colorada mine has come from two other sources than the glory hole mentioned above. One of these is an undercut block caving operation in the hillside near the mine ore bins. From a tunnel several hundred feet long extending through a saddle in a ridge, crosscuts were driven under the hill, and a block of ground undercut a short distance above the tunnel level by a network of small drifts and crosscuts. This ground is a well-fractured, highly altered, soft volcanic tuff. It caves readily and is mined cheaply. When placed in a stope it packs firmly, making an ideal fill. The third external source of fill for this mine is an old surface dump at the nearby Kirk mine. This is composed chiefly of well broken, soft, altered limestone, which likewise makes a very satisfactory stope fill. It is loaded into cars by a large electrically operated scraper. The comparative costs of the three waste fill operations are given in the following tabulation.

Details of gobbing cost, La Colorada mine, Cananea, Sonora

First six months, 1931. Costs in gold dollars per ton of gob.

	Glory hole[1]	Kirk dump[2]	Kirk block[3]	All[4]
Labor	$0.0725	$0.0395	$0.0599	$0.0552
Explosives	.0562	.0014	.0293	.0332
Timber	.0004	-	.0125	.0038
Miscellaneous supplies	.0029	.0057	.0022	.0028
Drills, etc.	.0023	-	.0011	.0013
Air	.0099	-	.0048	.0057
Electric power	-	[5].0084	-	.0014
Motor haulage	.0116	.0638	.0671	.0422
Total cost	0.1556	0.1198	0.1769	0.1456
Tons	54,073	20,035	36,710	124,970
Tons per shift	45.40	89.39	38.85	52.95

[1]Open glory-hole system, in hard porphyry (see U.S. Bureau of Mines Inf. Circ. 6247, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico, by William Catron). [2]Old mine dump--finely broken, decomposed limestone; scraping into cars. [3]Undercut block caving in a well-fractured, soft, altered tuff. [4]Several thousand tons was derived from underground development. [5]Power for electric hoist operating scraper.

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At Pilares a glory-hole system is in use, except that one of the three glory holes in operation is in reality an old mined-out and filled stoping area, in which the only blasting now required is to remove the old stope pillars. Because of the difficulty experienced from the stickiness of the fill after winter rains, provisions have been made to store large quantities of waste in old shrinkage stopes in the mine so that active glory-hole mining may be discontinued in large part during the winter.

Almost everywhere the glory-hole method of mining is subject to the unfavorable effects of rains or snow, which often force the stopping of mining, but in most respects it is the cheapest method of supplying large amounts of fill.

Old stopes supply satisfactory waste filling material under certain conditions. At the Eighty-Five mine the chief source of fill in the cut-and-fill stopes is material drawn from old caved shrinkage stopes. At this mine no damage results from allowing the stopes to cave after the level has been mined out. Similarly, at the Iron Cap, the fill from old stopes was drawn out for use in the stopes lower down after it became no longer necessary to maintain the levels above. The only cost in such cases is the cost of drawing and tramming, and occasionally the waste can be drawn directly from one stope into the other.

In a few mines stopes have been operated partly or wholly for the purpose of supplying fill. It is stated that part of the fill used in stoping at the Iron Cap mine came from shrinkage stopes above the ore zone. At Cananea a part of the production of a large shrinkage stope that was too low grade for ore was used for fill. The stope was being carried up through ground which was nearly barren close to the surface in order to make a cut-off to protect the shaft from subsidence due to mining.

Mill tailings are used sometimes for fill and, as stated before, have very desirable characteristics. At the mines of the Copper Range Co., Michigan, a certain amount of stamp sand was formerly transported to the mines by the same cars which carry the ore to the mill. The sand was dumped into long raises close to the hoisting shafts. At Matahambre[28] [29] tailings are pumped from the mill pond to a classifier at the mine, there deslimed, and run into the mine through pipes. Water is added to dilute the thickened sand pulp to about 66 per cent solids by weight.

All the foregoing methods derive waste from points outside the stopes. There remain three sources of waste fill which may be of great importance. Where sorting is done in the stopes a varying but sometimes considerable proportion of the fill is obtained from this process. Ordinarily, if two-thirds of the material broken can be used for this purpose, no outside source of fill is needed. At the Copper Range mines, as mentioned under the heading "Sorting," about half of the material broken in the stopes remains as fill.

In some mines, especially narrow-vein mines, the practice of shooting down the wall for filling is followed. Mining was done in the Smuggler-Union vein in the Telluride district by this method. Sometimes in such cases the vein and wall are shot together, but usually the weaker member is broken down first. At the mine of the Molybdenum Corporation of America at Questa, N. Mex., where molybdenum sulphide ore is mined from narrow and irregular stringers and the veins are usually not wide enough to give working room, the usual practice is first to blast down the necessary amount of waste, sort out of it such ore as can be seen, and then pick down the soft and friable ore by hand, or break it down by very light shooting. Shooting holes in the ore streak is avoided when possible because of the loss from scattering of the sulphide. At times, however, it is necessary to break down ore and waste together. It is estimated that 60 per cent of the material broken is rejected and left as fill in the stopes.

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28 - Richert, G. L., Filling Stopes with Mill Tailing at Minas de Matahambre in Pinar del Rio, Cuba: Eng. and Min. Jour., March 2, 1929, vol. 127, pp. 348-352.

29 - Richert, George L., Mining Methods at Minas de Matahambre, Pinar del Rio, Cuba: Inf. Circ. 6145, Bureau of Mines, June, 1929, 18 pp.

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In mining wide veins the practice of breaking the necessary fill from one or both walls would create a back too wide for safety.

The third source of waste within the stope is prospect drifts or inclined raises driven into the walls or ends of the stope, and horses of waste in the ore, which are blasted down after mining partly around them. Raises are generally enlarged at a short distance from the stope so as to break a maximum tonnage of waste with each hole drilled. In some cases they are allowed to cave and the caved rock is drawn out as long as needed. At the United Eastern, in the Tom Reed Extension workings, much of the filling was obtained from prospect raises in the walls. These raises were as long as 60 feet where the stopes were wide and needed large amounts of fill. It was found that this practice weakened the walls to such an extent as to cause much trouble, especially when the stope approached the level above; timber cribs then had to be used to support the back. At Butte the practice of crosscutting the walls of a stope to supply waste fill is likewise condemned on the ground that it weakens the walls. In that district most of the fill is from development work. However, the driving of openings into the stope walls may be desirable for the reason mentioned previously--that of finding parallel veins or branches of the main vein.

When waste is brought into a stope from outside, a grizzly is sometimes placed over the top of the fill raise to insure that all the rock will be under a certain size and to serve a safety purpose as well. Such grizzlies are made generally from old rails, with openings 9 to 12 inches wide. If the waste is to be spread by hand shoveling or by wheelbarrows or cars, it is especially desirable to eliminate bowlders. At United Verde, where a large part of the waste fill comes from power-shovel operations in the open pit and is dropped through long raises to the active mining levels, it is found that if the waste is kept drawn well down from the tops of the transfer raises the rock is crushed by dropping and issues from the bottom much finer than when put in at the top.

Spreading Fill

The waste can be spread in a stope by a number of methods: Gravity, hand shoveling, scrapers, wheelbarrows, or cars; for sand fill any of the above methods or water or compressed air may be used.

Gravity is the cheapest method of spreading, and in the ideal rill stope little or no labor is required for this operation. If the waste enters at the apex of two rills it may be necessary to build bulkheads to divert the waste to one side or the other. If flat places occur, some hand shoveling or scraping must be done. If the stope is wide, some labor will be required to spread the fill evenly across the width of the stope.

Hand shoveling is relied on usually to spread the fill in a horizontal cut-and-fill stope when the fill raises are close together. For distances greater than the economic hand-shoveling distance, wheelbarrows may be used, especially in narrow stopes where the tonnage to be moved would not justify installing cars or a scraper.

If the waste is to be moved a considerable distance from the waste raise in the stope, a common practice is to lay track and handle the waste in cars. In this event a loading chute is usually built at the bottom of the raise, and taken down and rebuilt after each slice mined. At Pilares, 2-wheeled conveyances like cement buggies, called "chalupas," are used for spreading fill from loading chutes under the waste fill raises. Cars are used in place of the chalupas where irregular spacing of the waste raises has resulted in long hauls.

Cars used in stopes are usually the regular small mine cars holding 16 to 20 cubic feet of rock. The track may consist of light rails, usually 18-inch gage, with wood ties spaced well apart. United Verde uses sectional track for tramming in the stopes. Using this track and the 18-cubic foot scoop-body car designed for this service, waste trammers average 70

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cars per shift. This would provide fill for 125 to 150 tons of ore produced daily. In the horizontal cut-and-fill system formerly used at the Champion mine of the Copper Range Co., tramming of both ore and waste was done in the stope car previously described. A rill cut-and-fill system is now employed in which the ore is handled largely by scrapers and the waste runs into place by gravity.

If scrapers are used to move the ore to the chutes, they are generally used to spread the fill also. This is the case at Cananea and at Park Utah. Moreover, at several mines where scrapers are not used on ore they are found advantageous for spreading fill. At Pilares scrapers are used in some of the large stopes; scrapers are used to some extent at Matahambre for spreading such development waste as is put in the stopes; scrapers were used to move the fill in the larger open stopes at United Eastern.

When mill tailings are used for filling they may be spread by compressed air, flooded in as a thick pulp, or handled dry in cars. At the Copper Range mines a system of spreading the stamp sand formerly used for filling consisted of a sand tank and a compressed-air jet.[30] This device apparently operated efficiently, but has been abandoned. At Matahambre, as stated, deslimed mill tailings are used. It enters the mine mixed with water as a pulp about two-thirds solids by weight, and flows by gravity through pipes into the stopes. After extensive trials with iron pipe and rubber hose, rubber-lined iron pipe was found most economical, though expensive in first cost. The sand is spread around the stope in a semifluid condition, low dams of sand being built up to restrict its travel if desired, and using pipe or hose to lead the flow from the fill raise to any point in the stope. Because of the range and flexibility of this system, fill raises are now spaced on 100 or 150 foot centers instead of the 80-foot intervals formerly used. To keep the sand from washing into the chutes or manways, these are wrapped with burlap before filling. The chutes and manways are also lined with plank, which prevents any large runs of sand in case the burlap rots away. Within 12 hours of the time the filling is run into the stope the water has drained out to the level below, leaving a smooth solid floor in the stope, which is apparently used to blast the next cut of ore upon without any flooring. A filling of this type will settle very little, and gives excellent support to the walls, as it washes into all cracks and irregularities. The cost of filling was given as 35.3 cents per ton for 63,354 tons of tailing placed in the mine during the first nine months of 1928. This compared very favorably with the cost of 55 cents per ton for rock mined at the surface and used for stope filling. Moreover, it was believed that with certain changes and with the experience gained in the first years of operation the cost would before long be reduced to about 20 cents per ton of filling placed in the stopes.

In horizontal cut-and-fill stopes the fill is spread to within 3 to 7 feet of the back. This distance depends partly on drilling practice. If stopers are used, as in a few cases, 6 or 7 feet is the least working space that should be left under the ore. If breast stoping is practiced, the only work that has to be done under the old back is the spreading of the fill and laying floor. Placing the fill close to the back reduces to a minimum the height of the back over the miners and muckers after the next cut, and correspondingly increases safety. If, however, under such conditions a large or thick round is broken, a considerable amount of mucking will have to be done before drilling can proceed, as the muck pile will block off the face. If it is desired to have the miners drilling and breaking continuously and without any interference from mucking operations, it may be better to leave enough room between the fill and back to accommodate most of the broken ore, which will then form a working platform at the right height for setting up a drill column (see fig. 6).

In an inclined cut-and-fill stope the fill is ordinarily placed as close to the back as men can work to lay the floor, that is, 2 to 4 feet. In these stopes the face is cleared of

30 - Schact, W. H., Mining Methods of the Copper Range Company: Trans. Am. Inst. Min. Eng., vol. 72, 1925, p. 363.

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ore by gravity after each round, unless the ore is deliberately allowed to accumulate to form a working platform for drilling.

Filling in a stope settles slightly after being placed. In rill stopes at the United Verde it is observed that this settling has a differential effect which tends to flatten the pitch of the rill from its original slope of 40° to about 37° by the time a cut is finished. Such settlement is due in part to the weight of the fill itself and partly to the impact of ore falling on it. It may cause some difficulty in keeping cribs or stulls tightly blocked against the back but will seldom be enough to allow any appreciable movement of the hanging wall, such as might result from the failure of timber in a square-set stope, or the motion of ore in a shrinkage stope.

After being placed, the fill is often wet down to help consolidate it. When soft, fine material is used, this is effective and produces a smooth, hard floor on which the loss of fine ore will be small.

At Butte an extensive hydraulic filling operation was carried on in a section of the Anaconda property where stubborn mine fires were interfering with mining.[31] From 1917 to 1921 about 460,000 tons of solids (mill tailings) had been used for this purpose, and it was estimated that about 800,000 would be used in all. The material was tailings from a concentrator; it was passed through spitzkasten to thicken it to about 25 to 30 per cent solids. The material was much finer than that used at Matahambre, consisting of over 50 per cent minus 200 mesh, whereas the latter contained only 8 per cent. Nevertheless, it apparently presented no difficulties in handling or in draining in the stopes, and formed a compact fill which would stand in a vertical face when excavated. Apparently no difficulty was experienced with pipes wearing out, which was one of the greatest troubles encountered at Matahambre. Six-inch, extra-heavy, cast-iron, flanged pipe was used in the shafts and 4-inch cast-iron or wrought-iron pipe on the levels. It was found that good practice permitted 800 feet of horizontal transportation for each 100 feet of head in the shaft lines.

The cost of procuring and placing waste in a stope is a direct charge against ore extraction, just as properly as timber and timbering labor would be in a square-set stope. Under certain conditions this cost might be so high as to give preference to some other means of support. At a quicksilver mine in California, where some underground mining is done by the square-set method, the timber is used merely for a working platform and perhaps to provide good temporary protection for the miners. The walls of the vein are so strong that after a stope is mined out the timber is often removed and used over again in another stope. Under such conditions timbering is probably less expensive than filling would be. As a further illustration, it is well known that the cut-and-fill method was introduced into many mines, as in the Michigan copper and iron mines and especially in the copper mines of the Southwest, only when the exhaustion of local timber and the cost of imported timber made the square-set method over costly. Whenever timber is a satisfactory support and is cheaper than waste fill its use should be considered.

In considering the cost of filling, the relative densities of ore and fill must be kept in mind and the cost translated preferably into cost per ton of ore. Filling of low density will ordinarily be more cheaply handled as well as occupy more room per ton. The cost of filling as well as the cost of breaking may be decreased by mining thicker slices.

MODIFICATIONS AND COMBINATIONS OF CUT-AND-FILL WITH OTHER STOPING METHODS

The use of square-setting for the removal of pillars under and over the levels has been mentioned. This, however, scarcely comes under the heading of combined methods, as so small

31 — Rahilby, H. J., Mine Fires and Hydraulic Filling: Trans. Am. Inst. Min. Eng., vol. 68, 1922, pp. 61—72.

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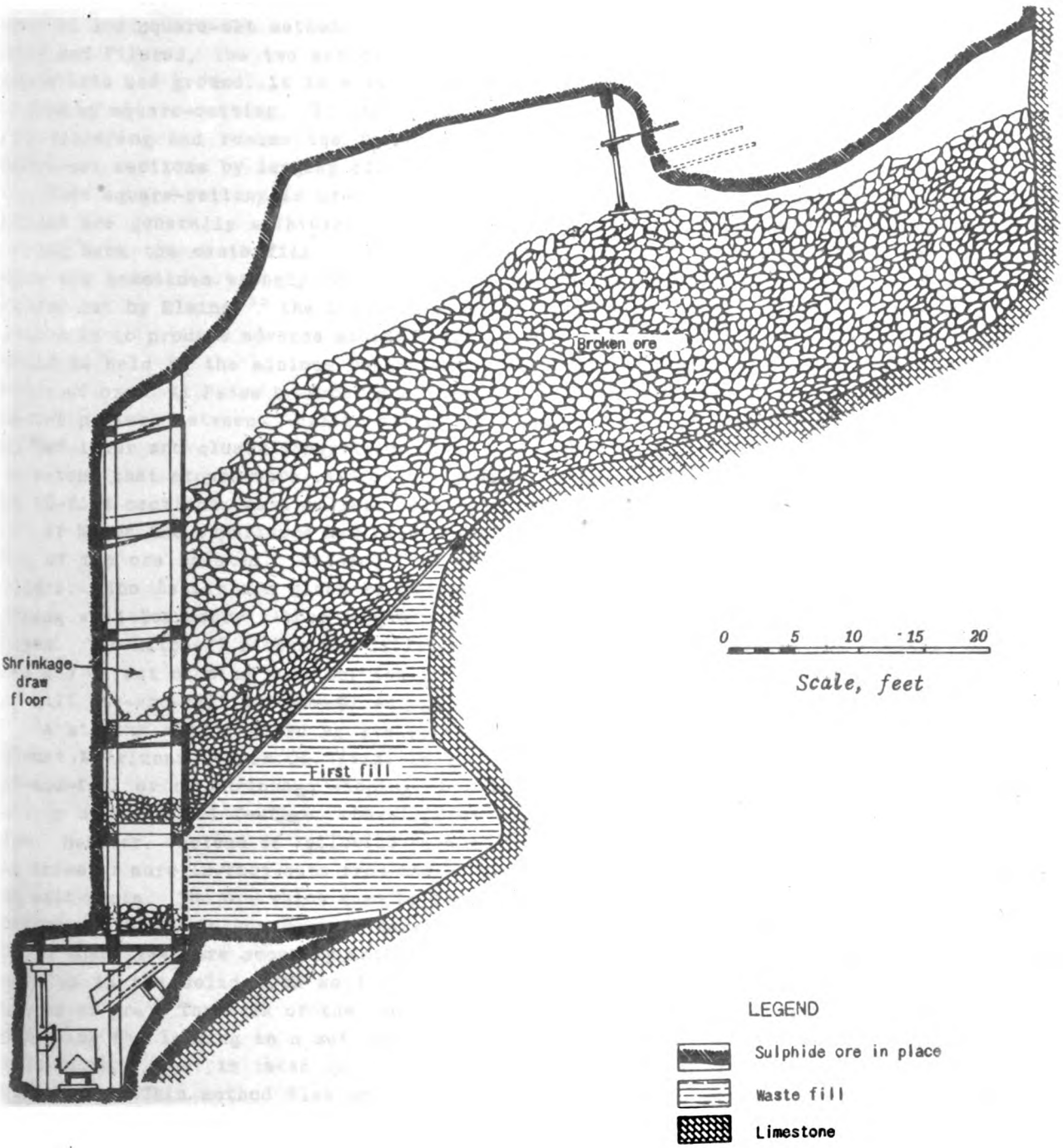

Figure 20.- Semishrinkage stope, Campbell mine

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a percentage of the ore is extracted by the square-setting. A true combination of the cut-and-fill and square-set methods is not uncommon, however. At the Pecos mine, as at United Verde and Pilares, the two methods are used interchangeably: If a cut-and-fill stope advances into bed ground, it is a very simple matter to level the fill, stand square-sets and proceed by square-setting. If conditions revert to the former status, it is equally easy to stop timbering and resume the former method. Chutes are readily maintained through the square-set sections by lagging off the desired sets.

When square-setting is used to remove the pillars between or under filled stopes conditions are generally unfavorable because of heavy, fractured ground and the necessity of holding back the waste fill in the old stope. This is reflected in higher mining costs, which are sometimes wrongly believed to be due to the use of square-setting. However, as pointed out by Elsing,[32] the inevitable result of cutting a block of ground into stopes and pillars is to produce adverse mining conditions in the pillars. Pillar mining, accordingly, should be held to the minimum consistent with safety and with the reasonably complete recovery of ore. At Pecos a former practice was to mine stope sections 15 to 60 feet long with 15-foot pillars between. Pillar mining was found to be more expensive and to require more skilled labor and closer supervision than the stope mining. Therefore, it was abandoned to the extent that stopes were laid out in panels 100 to 150 feet long, to be mined in successive 15-foot sections, with pillars left only at the panel boundaries.

At Magma the experience has been different, and pillar mining furnishes roughly 40 per cent of the ore hoisted. Cut-and-fill stope sections 16 feet long alternate with 14-foot pillars. The latter are mined by the Mitchell slice system, which consists of underhand stoping with temporary timber support of the walls and of the fill in adjacent worked-out stopes. The Mitchell stopes are filled as each is completed for permanent support. At this mine the direct stoping cost per ton in the pillars is only very slightly more than that in the rill cut-and-fill stopes--$1.46 against $1.35.

A stoping method known as semishrinkage stoping is used at the Campbell mine of the Calumet & Arizona Copper Co. (fig. 20). This may be considered a modification of either cut-and-fill or of shrinkage stoping, but is closer to the former because permanent waste fill is an essential feature. Development is the same as for the usual rill stope at that mine. However, instead of following each mining slice by filling, the cycle consists of mining three or more 12-foot cuts from the back of the stope, then drawing the ore out and filling with waste. The ore which must be removed during mining because of the "swell" is passed through grizzlies in square-sets carried up at one end of the stope. As soon as they are put in these sets are securely anchored to the wall by cables and short pieces of steel set in holes in the solid rock so that they will not collapse when the stope is temporarily emptied of ore. The bulk of the ore is drawn out, after a careful barring down of the back, by opening the lagging in a set near the bottom of the ore. After the ore is cleaned out, the temporary floor is taken up, the stope filled, and the flooring re-laid as in a regular rill stope. This method ties up a considerable amount of broken ore, but not as much as ordinary shrinkage stoping. It gives no opportunity for sorting, and is somewhat erratic in ore production and fill requirements, but the costs of mining and filling are reduced by putting these operations on a larger scale, and the cost of flooring is likewise reduced.

A modification of cut-and-fill stoping, which might best be called underhand cut-and-fill, is used in the Tiro General mine at San Luis Potosi, Mexico.[33] The complex sulphide orebody is a vein dipping about 70° and averaging 30 feet wide. The footwall is regular and

32 - Elsing, M. J., Square-Set Stoping: Eng. Min. Jour., vol. 130, Oct. 9, 1930, pp. 329-332.

33 - Willey, Howard, Top Slicing with Filling of Slices, as Used at the Charcas Unit of the Cia. Minera Asarco, S. A.: Tech. Publication 364, Am. Inst. Min. Eng., 1930.

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well defined by post-mineral faulting, which has produced a smooth surface frequently covered with a slippery clay gouge. The footwall is limestone, strong in places, but in other places broken and altered to such an extent as to require heavy timbering. The ore is fractured and jointed, both transverse to the vein and in horizontal or flat-dipping planes. The latter system is pronounced, and is responsible for the ore's being very difficult to support. The hanging wall is an irregular surface, frequently an economic assay limit, so that stoping widths range from a few feet to 90 feet. The hanging-wall rock is a shattered and altered monzonite porphyry which almost always requires heavy timbering. The upper levels of the mine, in which considerable ore remained, were honey-combed with old workings, the stopes of which were open, caved, or filled with waste or formerly waste material. It was necessary while extracting virgin ore from the lower levels to avoid wrecking the upper part of the mine or causing any subsidence which would affect the shaft or surface plant. It was also desired to develop a method which could later be used for mining in the upper portion of the vein.

The method finally adopted consists of mining 11-foot horizontal slices in downward progression from 100-foot blocks, flooring and filling promptly to sustain the weight of the walls, and temporarily carrying the weight of the fill on stulls and flooring while mining out each slice (fig. 21). The chief advantages of the method are safety to workmen, complete extraction of ore, and lower costs than are obtained with overhand square-setting and filling, which was the only other method which had proved applicable under the very adverse conditions noted. The saving in cost was 48 and 36 cents per short ton when working in virgin ground and in old workings, respectively, and was effected chiefly in decreased timber consumption. The production in tons per man-shift was about the same as for square-set stoping:

	Tons (2,000 pounds) per man-shift	
	In virgin ore	In old workings
Square-setting	0.70	1.37
Filled top-slicing	0.59	1.48

In 1929 or 1930 a small orebody was mined at the Copper Queen by an unusual combination of square-setting and inclined cut-and-fill stoping. The ore was outlined on an intermediate level by an untimbered drift about 100° feet long, and two or three short crosscuts. A 2-compartment raise extended from the drift to the level 70 feet above. The drift was first timbered with sill floor square-sets, having 9-foot posts. Chutes were built at convenient intervals, and a single row of square-sets was carried up over the drift, extending both vertically and lengthwise to the ore limits. This formed a timbered slot one set wide, over 125 feet long, and about 55 feet high. The ore did not extend to the level above. The ground was hard, heavy, and blocky. Square-setting, however, seemed an unnecessary expense, and cut-and-fill stopes were laid out normal to the "slot," 20 feet wide, with 15-foot pillars between. These stopes were mined as rills, running the broken ore down to the "slot" by gravity, and introducing fill through small untimbered raises run up at the far end of each stope section. Upon reaching the top of the ore the pillars between stopes were mined downward, using underhand square-setting.

Drilling and breaking efficiencies were very high in the first or cut-and-fill part of this work. Having a "slot" across the ends of the stope sections, instead of separate two or three compartment manway and chute raises for each stope, made access easy for men and supplies and provided a convenient place to break bowlders and draw off the ore. However,

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considering the difficulty and cost of extracting the badly crushed and heavily loaded pillars, it is uncertain whether the method used or straight square-setting would have produced the better results. The average labor efficiency for the whole orebody was about 4 tons per man-shift, which is lower than the usual efficiency in square-set stoping at this mine.

At the Gilman property of the Empire Zinc Co.[34] certain zinc orebodies are mined by stopes and pillars, the stopes being worked by a modified cut-and-fill method. The orebodies are pipe-like, with transverse vertical sections perhaps 50 feet high by 150 or 200 feet long. They are laid out into transverse stopes and pillars, the stopes being 35 and the pillars 15 feet wide. The stopes are mined by horizontal cut-and-fill, but a row of square-sets is carried up on either side, as the stope advances. This timbering supports the back, which seldom stands satisfactorily over a full 35-foot span. The square-sets likewise provide a secure gob fence, and serve later as support for the long timbers used in stoping the pillars, which are worked by the Mitchell slice system. The upper one or two 8-foot slices in the stopes generally are square-setted, as the ground is weak near the top of the ore. Thus it is apparent that 25 to 50 per cent of the cut-and-fill stopes are actually square-setted. Both ore and waste are generally transported in the stopes by scrapers. Comparative statistics follow, in which regular square-set stoping is contrasted with the two phases of the stope-and-pillar method:

	A	B	C
Tons per machineman shift	25.7	28.4	21.7
Tons per timberman shift	13.5	21.4	18.4
Tons per mucker shift	13.5	19.5	19.3
Tons per fill-labor shift	35.1	44.2	-
Tons per total man-shift	4.64	6.4	6.4
Timber, board feet per ton	13.9	7.3	6.7
Powder, pounds per ton	0.67	0.53	0.41

Column A gives the average of 11 square-set stopes. The powder consumption shown includes powder used in blasting the back of the stope for filling. Mucking and filling are done by hand. Column B is the average of 9 cut-and-fill stopes, including, however, the square-setting of the tops of stopes. Mucking and filling are largely done by scraping. Column C is the average of only two pillar stopes, which have not yet been filled. Hence, the tonnage is not sufficient for accurate comparison. Mucking was by hand, but most of the ore went to the chutes by gravity.

STOPE-CREW ORGANIZATION AND METHOD OF PAYMENT

The organization of stoping crews is so closely related to the method of payment that the two subjects are most conveniently considered together. Generally the shift boss whose territory includes a given stope has direct authority over the work in that stope. In some cases, however, he is assisted by a stope boss, as at Cananea, for example, where a stope boss generally supervises the work in two or three stopes. If the contract system is in force, the head contractor may in effect perform the duties of a stope boss. He may thus be in charge of three to a dozen men. The stope boss sometimes works as a miner in addition to his supervisory duties. If no man in the stope is officially named as the boss, the miners usually assume the responsibility and direct the work of the muckers or trammers.

34 - Borcherdt, W. O., and Technical Staff of Company, The Empire Zinc Company's Operation at Gilman, Colorado: Eng. and Min. Jour., vol. 132, August 10, 1931, pp. 99-105.

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No apparent trend is discoverable in the mines discussed in this paper either toward or away from specialization in work, although the nature of the changes made in some mines and the great differences noted in practice would indicate that the subject has received considerable thought. Some mines are changing toward the generalization of the duties of individual miners while other mines still maintain the older system of specialized miners, timbermen, muckers, trammers, repairmen, and so on. At Cananea, Pilares, Magma, Park Utah, and the Ground Hog, for a few examples, the work in the stopes is done more or less indiscriminately by all the men, the sole distinction being that most of the shoveling is done by muckers or helpers. At the Ground Hog mine it was found that the separation of drilling and timbering resulted in loss of coordination; often the miners broke ground so far ahead of the timbermen that caves occurred. Under the present system the same men drill and blast, timber, and muck if need be. The stope crew consists of one timberman, who is the highest paid man, one machineman, and one helper. The change in system has had favorable results. In the copper mines of Michigan it was similarly once the practice to employ specialists: Machinemen, timbermen, dry-wall and mill-hole builders, fillers, and pickers. It was difficult to secure satisfactory cooperation between the separate crews, and considerable lost time occurred because of one group of men having to wait for the other. Now each stope is worked by one miner and one or two helpers or muckers who perform all the necessary operations, and a considerable gain in labor efficiency has resulted from the change.

At the Champion mine, in this district, the stope contract system is designed to encourage careful sorting of the ore. A miner and one or two muckers constitute a stope crew to break the ground, sort and tram the ore, and place all necessary props to support loose ground. Miners receive 45 cents a day more on company account than do the pickers, but the bonus payment is divided according to shifts worked by each man. Each month the engineers measure the stope to determine the volume broken and the average width. Underground tramming records and the time books supply the additional information needed to calculate the bonus payments. Three separate items comprise the total bonus payment. The first is a bonus based on tonnage broken, modified by the width of stope; the wider the stope the less the bonus for a given tonnage per man-shift. The second item is based on the tons of ore produced and the sorting ratio; the more sorting is required the higher the bonus paid per ton of copper rock shipped from the stope per man-shift. The third item is a powder allowance based on the tons broken and the width of the stope.

The United Verde mine offers a good example of the principle of specialization of work that has been applied to such a marked degree in the large mines using the undercut block caving system of mining as well as in most modern manufacturing plants. At this mine, in a cut-and-fill stope with an area of 5,000 to 9,000 square feet, which may produce about 150 tons per day (two 8-hour shifts) the usual crew on each shift consists of one miner, who drills, loads and blasts the round; about six muckers; one pluggerman, whose duty is to drill and blast bowlders too large or hard to be broken by hand; two to four timbermen, who build chute and manway raises, construct grizzlies, and build waste pockets; two more timbermen, who repair chutes and other timber structures; and one or two waste trammers, who tram and spread the waste fill--a total of 13 to 16 men. A labor efficiency of 8 tons per man-shift in horizontal cut-and-fill stopes is given for the year 1929. Mining, mucking, waste filling and repair timbering in most of the stopes are done on the bonus system, men being paid according to the ratio of their performance to a standard set at the beginning of each bimonthly period. New timbering is on contract and is paid for at so much per timber crib placed, or per waste pocket built.

Recently the contract system has been introduced in the stopes in some small sections of the United Verde. All labor and explosives are charged against a given contract, and at the end of each month any contract earnings, based on the tons of ore extracted at the given

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price per ton, are divided among the men, miners receiving double what the muckers receive. The main tangible advantage to the company has been in a saving in explosives. However, a noteworthy psychological advantage has resulted from the change, in that the men are more contented. They feel that they are receiving the full contract price per ton and are the chief gainers from any extra exertion or improved efficiency, rather than having to divide the rewards with the company. The bosses, moreover, tend to give closer supervision to stopes which are worked on contract and keep more accurate accounts than under the bonus system. Some time ago it was attempted to establish a powder bonus in stopes working under the bonus system, in order to decrease the explosives cost. This was unsuccessful because of its unfavorable psychological effect on both the men and the bosses.

At Pilares an elaborate system of stope contracts has been developed during a long period of operation. A given crew of men performs all operations in a stope. One contract price covers breaking, shoveling, and timbering. Each other operation has a more or less standard price and involves extra payment. It is found that orders pertaining to a change in method or arising from safety considerations in the stope are much more willingly and efficiently carried out when the men know that they will be paid for the additional labor than when the extra work is all included in the single main contract.

In almost all of the mines discussed in this paper, all or a part of the stoping is done under either bonus or contract systems, with the latter predominating. The contract system, in which every unit of work is paid for at a flat rate, offers the greatest chance for large earnings to the exceptionally able workman, whereas the company does not benefit so directly by decreased unit costs. Contracts are, therefore, more difficult to set at the correct figure. Such bonus or contract agreements almost always guarantee at least day's wages to the men. Frequently the contracts are verbal only and seldom are written for any definite amount of work even when signed agreements are drawn up. Most companies have found it advisable to give the men the advantage in rate changes during a contract period, with the result that prices are never lowered during a period but are usually raised if it is found that a mistake has been made in setting them.

VENTILATION

Ventilation of cut-and-fill stopes is usually by upward air currents which enter by way of the manways from the extraction drift and reach the level above through manways or fill raises. In a horizontal stope a fill raise is likely to be choked with waste part of the time. If only one raise exists, it should have two compartments, one of which can be kept open for ventilation. Ore chutes are unsatisfactory air passages, as their usefulness is intermittent and ventilation through them may have to yield precedence to ore handling.

Ventilation in most stopes is concerned primarily with removing the smoke and gas from blasting. In the massive sulphide stopes of the United Verde mine this is of vital importance, as blasting in this ore produces particularly noxious gases. At a few mines, such as Magma and the Eighty-Five mines, the supplying of cool, dry air to permit men to work is probably the most important function of ventilation.

SAFETY

Accident hazards and accident prevention in cut-and-fill stopes have many features in common with other types of mining; these features will not be discussed here, but attention will be concentrated on some which are more or less peculiar to this method of mining.

The cut-and-fill method of stoping is a safe method when rightly used under proper conditions. Both miners and muckers are working a large part of the time under a rock back

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from which large slabs or bowlders may drop, and which must always, at least, be watched carefully and kept trimmed to prevent the occasional falling of small pieces of rock. In a shrinkage stope it is often possible for the men to avoid the center of the stope by drilling long holes from near the edge, and there is no need for muckers in the stope, so that fewer men are exposed to the danger. Open stoping is used only in the best of ground, and after the first thorough cleaning of the back there is seldom any falling of rock. In timbered stopes the men can be protected by lagging overhead. Therefore, the first rule of accident prevention in a cut-and-fill stope should be "bar down."

A horizontal cut-and-fill stope usually has a number of chutes and two or more manways opening at the level of the floor of the stope; a hazard exists of men falling down these openings. Chute grizzlies and manways both should be planked over when not in use. Manways may be guarded by railings or other devices. If it is practicable, the use of one raise for both manway and ore chute should be avoided. If two raises are not available, the danger that ore will be shoveled or pushed into the manway must be guarded against by covering the manway tightly with planks, or by means of a trap door, or by building the manway side higher than the chute. Figure 14 shows a satisfactory chute and manway combination used at Magma, which need not necessarily have two chute compartments.

In a rill stope there is the hazard of rolling bowlders. This danger increases with the steepness of the rill and the size of the stope. If the rill is steep enough to cause the ore to roll or slide to the bottom, which is the essence of the method, the muckers should if possible be given protection in the form of square-sets or cribs placed next to the grizzly at the bottom of the slope. This is the practice at Magma, Pilares, and Calumet & Arizona. At the Ground Hog mine the use of a shoveling platform makes it possible to place stulls or other stope timber to support the back, and the level space gives a safe and convenient working space for the muckers.

Fill raises are a potential accident hazard in cut-and-fill stopes because of the possibility of rock fill being dumped in without warning. If temporary loading chutes are used, this danger is eliminated except during the period when the chute is removed and a new cut is being mined adjacent to the raise. At such times, and in any stope where the fill raises are left open, extreme caution must be used by the filling crews and the miners themselves. Many accidents have occurred from dumping fill into the wrong raise, or from lack of contact between the stope crew and the waste trammers on the level above.

COSTS

The following cost tables, 4, 5, and 6, have been compiled from Bureau of Mines Information Circulars and from the other sources referred to previously.

Table 4, which gives mining costs in units of labor, power, and supplies, includes a number of mines which use the cut-and-fill method only in part. The costs for these mines, therefore, apply to no one method, except that at United Verde the figures were segregated according to stoping method and, therefore, apply to cut-and-fill mining only. As development cost at United Verde was given separately, a proportionate part of this was added to the mining cost to give the figures shown.

Table 5, giving mining costs in dollars and cents, must be interpreted with the same understanding of the methods used at the various mines.

Table 6, showing stoping costs by the cut-and-fill method, gives figures which are free from the effect of combined stoping methods. For the four large mines, Campbell, La Colorada, Pilares, and Champion, the costs range only from $0.80 to $1.28 per ton. The costs at United Eastern and Iron Cap, however, were $2.92 and $2.90, or more than three times the lowest cost.

Because of the dissimilarity of mining conditions, no averages have been calculated. Each cost should be considered in connection with the operating problems and conditions at the mine in question.

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Table 4.- Mining costs in units of labor, power, and supplies at mines using the cut-and-fill method

Mine	Location	Date	Tonnage	Method	Labor, man-hours per ton							Tons per man-shift underground
					Breaking	Timbering	Mucking	Haulage, hoisting	Supervision	General	Total	
ited Verde[1]	Arizona	[2]1929	[3]900,000	61 per cent horizontal	0.235	0.466	0.623	0.238	0.126	1.037	2.724	2.92
				3 - do. rill								
				27 do. square-set								
				8 do. top-slice								
apa	Arizona	1928	263,094	Mostly rill-and-pillar	(	2.208	)	.639	.479	.934	4.261	1.83
				Some timbered-rill								
a Colorada	Mexico	[4]1929	262,278	40 per cent rill	.876	.799	.990	.494	.297	.163	3.619	2.21
				14 do. horizontal								
				Some square-set and shrinkage								
lares	Mexico	1929	835,036	59 per cent horizontal								2.11
				18 do. rill								
				18 do. square-set								
l Tigre	Mexico	[5]1924	6,080	Chiefly horizontal and inclined; some shrinkage and open								.64
cos	N. Mex.	1929	217,121	40 per cent horizontal	.917	1.476	1.015	.592	.071	.631	4.702	1.41
				50 do. square-set								
ricus	Michigan	1927	400,000	Horizontal and inclined								2.69
tahambre	Cuba	1928	364,746	70 per cent horizontal	1.07	[9].78	1.43	.64	.11	.28	4.31	1.86
				30 do. square-set								
ruma	Ecuador	1922	[11]93,000	Horizontal and inclined							16.28	
esta	N. Mex.	[12]1930	6,169	90 per cent horizontal								[13].66
				10 do. open								
mpion	Michigan	1921	531,780	Horizontal and inclined								2.56
mpion	Michigan	[15]1930	193,597	Rill stopes	.73	.30	.92	.86	.12	.11	3.04	2.64

- United Verde costs apply to cut-and-fill mining only, but have been refigured to include a proportionate share of development cost.
- First 10 months of year.
- Estimated on basis of 3,000 tons per day.
- First 6 months.
- November.
- Includes filling.
- From Mines Handbook, 1924 volume.
- June-November, 6 months.
- Includes surface labor.
- Last 5 months.

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Table 4.— Mining costs in units of labor, power, and supplies at mines using the cut-and-fill method Continued

Mine	Labor, per cent total cost	Explosives pounds per ton	Timber, board feet per ton	Power, kilowatt-hours per ton						
				Compressed air	Haulage	Hoisting	Pumping	Ventilation	Other	Tota
United Verde[1]	-	0.84	7.48	10.15	-	2.35	0.26	3.07	0.61	-
Magma	56	.90	19.00	14.16	-	8.92	5.54	4.91	1.08	34.6
La Colorada	52	1.37	7.19	6.0	1.0	3.4	.5	.2	.2	11.3
Pilares	-	.38	5.71	-	-	-	-	-	-	6.2
El Tigre	66	-	-	-	-	-	-	-	-	-
Pecos	75	.73	[6]10.00	6.88	.78	1.30	2.73	-	.09	11.7
Various	-	-	3.22	-	-	-	-	-	-	[7]8.
Matahambre	60	1.02	[10]1.15	8.00	-	3.00	1.79	1.22	-	14.0
Zaruma	-	2.24	-	-	-	-	-	-	-	-
Questa	77	2.98	[14]6.5	16.9	-	-	-	-	-	16.
Champion	73	.68	[8]1.87	2.56	.64	4.03	.79	-	2.56	10.
Champion	66	1.3	4.3	12.4	-	4.00	4.7	(1.0)		22.

1 — United Verde costs apply to cut-and-fill mining only, but have been refigured to include a proportionate share of devel ment cost.

6 — Given as 3.9 linear feet, of 8-inch average diameter.

7 — Plus 70 pounds of coal for steam power.

8 — Plus 0.002 cord of lagging.

10 — Plus considerable round timber.

14 — Stated as 4.86 linear feet of 5-inch round.

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Table 5.- Mining costs

Mine	Location	Date	Tonnage	Method	Labor	Super-vision	Explo-sives	Timber	Power	Com-pressed air, drills, and steel	Other	Total
United Eastern	Arizona	1917-24	717,582	Horizontal cut-and-fill	($2.628)		$0.513	$0.462	$0.289	-	$0.511	$4.403
La Colorada	Mexico	[1]1929	262,278	40 per cent rill	$1.102	$0.171	.279	.257	.070	$0.235	.328	2.441
				14 do. horizontal plus square-set and shrinkage								
Pilares	Mexico	1929	835,036	59 per cent horizontal	1.374	.219	.231	.319	.055	.232	.170	2.600
				18 do. rill								
				18 do. square-set								
El Tigre[2]	Mexico	1924	6,080	Chiefly horizontal and rill; some shrinkage and open	3.71	.32	.48	.51	.10	.56	.39	6.07
Pecos	N. Mex.	1929	217,121	40 per cent horizontal	3.201	.107	.259	.422	.091	.141	.677	4.893
				50 do. square-set								
Questa	N. Mex.	1929-30	17,959	90 per cent horizontal	(8.98)		(		2.63		)	11.61
				10 do. open								
Various (copper)	Michigan	1927	400,000	Horizontal and rill	(1.877)		(.621)		1.91	-	-	2.689
Champion	Michigan	[4]1930	193,597	do.	1.649	.107	.172	.109	.192	.197	.102	2.528
Matahambre	Cuba	1928	364,746	70 per cent horizontal	1.433	.155	.200	.269	.087	.161	.264	2.569
				30 do. square-set								
Zaruma	Ecuador	1920-22	270,000	Horizontal and rill	-	-	-	-	-	-	-	4.15

1 - First six months.

2 - Costs for November, 1924. Stated originally in Mexican currency and recalculated in this table to United States currency on the basis of $0.535 gold per $1.00 silver (Mexican).

4 - Last five months.

Table 5.- Mining costs-Continued

Mine	Development	Stoping	Haulage	Hoisting	General underground	Pumping	Surface expense applicable to underground mining	Total
United Eastern	$0.518	$2.923	-	$0.275	$0.248	$0.052	-	$4.403
La Colorada	.561	1.203	($0.166)		.278	-	$0.233	2.441
Pilares	.399	1.370	(.370)		.250	-	.211	2.600
El Tigre[2]	1.31	2.70	$0.67	.19	.34	.17	.69	6.07
Pecos	1.218	2.396	(.711)		.385	.178	[3].010	4.898
Questa	6.10	5.51	-	-	-	-	-	11.61
Michigan copper	.598	(1.765)		.158	?	.180	-	2.780
Champion	.490	1.163	(.552)		.078	.133	.112	2.528
Matahambre	.711	1.267	(.388)		.184	-	.019	2.569
Zaruma	-	-	-	-	-	-	-	4.15

2 - Costs for November, 1924. Stated originally in Mexican currency and recalculated in this table to United States currency on the basis of $0.535 gold per $1.00 silver (Mexican).

3 - Assaying only, other surface expense being distributed.

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Table 6.- Cut-and-fill stoping costs

Mine	Location	Date	Method	Tonnage	Labor and supervision	Explosives	Timber	Compressed air, drills, and steel	Power	Other supplies	Total
United Verde	Arizona	1929	Horizontal cut-and-fill	[1]600,000	[2]$0.85	-	-	-	-	-	-
Do.	do.	1928	Rill	[1]30,000	[2].71	-	-	-	-	-	-
United Eastern[3]	do.	1917-24	Horizontal cut-and-fill	717,582	1.750	$0.414	$0.445	-	$0.136	$0.178	$2.9
Iron Cap	do.	1923	Rill	85,211	1.20	.24	.22	$0.41	-	.82	2.9
Campbell	do.	1926-29	Rill No. 68	51,970	.620	.135	.155	.215	-	-	1.1
Do.	do.	1928-29	Rill No. 90	20,894	.70	.10	.13	.20	-	-	1.1
Do.	do.	1928-29	Mostly rill[4]	280,527	.765	.12	.175	.22	-	-	1.2
La Colorada	Mexico	[5]1931	Horizontal cut-and-fill	48,953	.256	.102	.114	.099	-	[6].227	.7
Do.	do.	[5]1929	Horizontal cut-and-fill	34,060	.500	.133	.254	.185	-	[6].170	1.3
Do.	do.	[5]1929	Rill	17,314	.494	.140	.284	.185	-	[6].170	1.2
Pilares	do.	1929	Horizontal cut-and-fill	[7]500,000	-	-	-	-	-	-	1.0
Do.	do.	1929	Rill	[7]150,000	-	-	-	-	-	-	.9
Pecos	New Mexico	1929	Horizontal cut-and-fill	81,827	1.52	.14	.26	-	-	.09	2.
Eighty-Five	do.	1928-30	Rill No. 10	7,034	1.26	.28	.20	.60	-	.03	2
Do.	do.	1929-30	Rill No. 26	4,056	1.62	.17	.27	.38	-	.02	2.
Michigan copper mines	Michigan	1927	Horizontal cut-and-fill[8]	400,000	1.340	([9].405)	-	.009	-		1.
Champion	do.	[10]1930	Rill	193,597	.851	.103	.061	.128	-	.020	1.

1 - Sixty and 3 per cent, respectively, of estimated annual tonnage of 1,000,000. 2 - Min. Cong. Jour., vol. 16, April 19 p. 320. 3 - Obtained by combining separately stated costs of stoping and filling. 4 - Entire Campbell area, including a m tonnage from other types of stopes. 5 - First six months. 6 - Filling cost. 7 - Fifty-nine and 18 per cent, respectiv of 1929 total tonnage of 835,036. 8 - Including small amount of rill stoping. 9 - 0.405 includes all supplies. 10 - five months.

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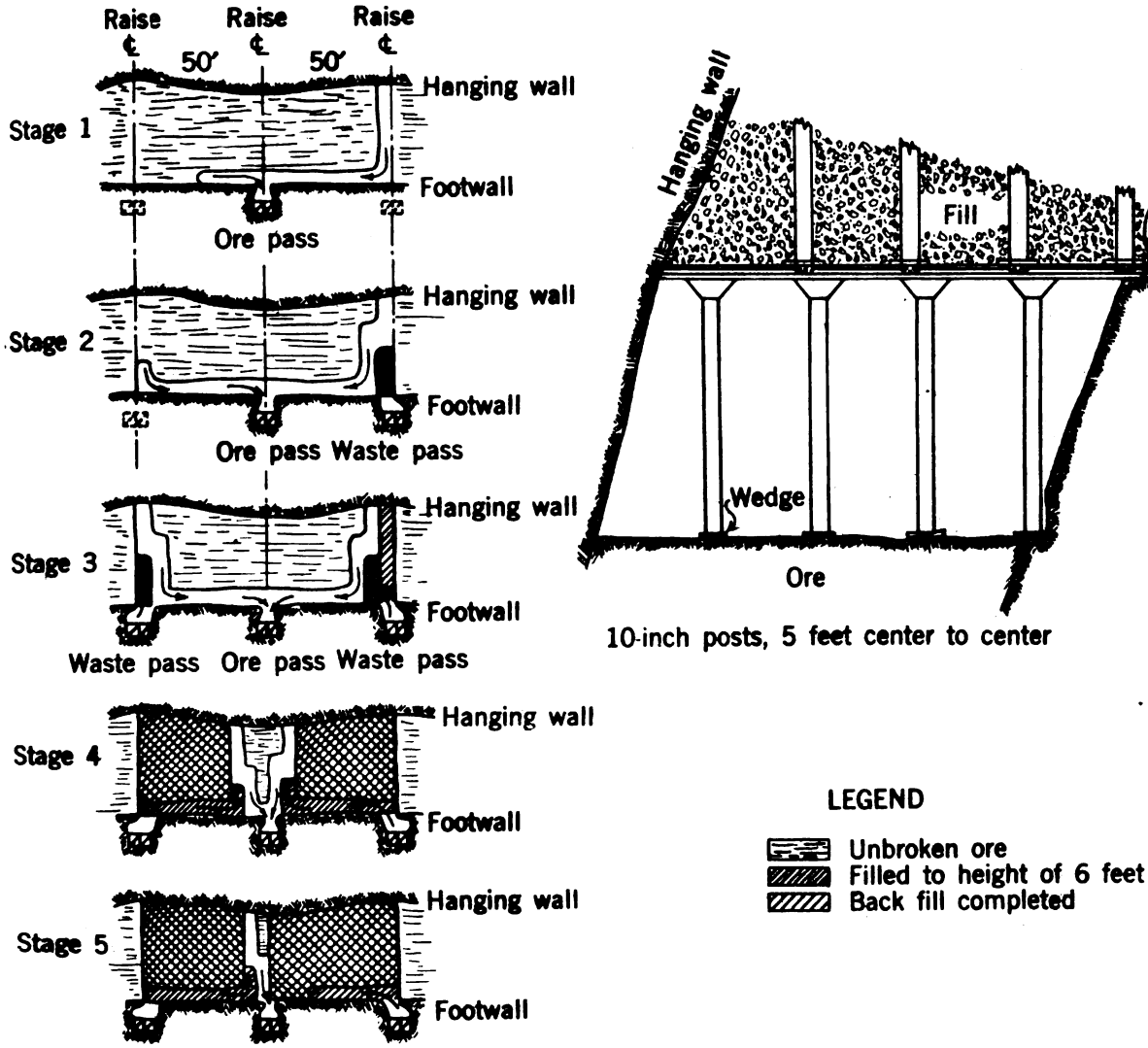

Figure 21.—Stoping method at Tiro General mine, San Luis Potosi

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